FROTH FLOTATION CHARACTERIZATION AND PROCESSING PLANT DESIGN FOR THE PLATINIFEROUS AND AURIFEROUS MARINE SEDIMENTS OF SOUTHWESTERN ALASKA

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A

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Abstract

The purpose of this study was to characterize, and investigate the beneficiation of, the platiniferous and auriferous marine sediments of Southwestern Alaska, located near Platinum, Alaska. The majority of placer gold particles are contained in the 50 × 150 mesh size fraction, while the platinum is finer, residing in the 100 × 200 mesh size fraction. Liberated placer gold and placer platinum group metals (PGM) particles are visible to the naked eye and readily observed under a binocular microscope. Preliminary, qualitative microprobe analysis of PGM grains from the flotation concentrate showed grains of nearly pure iridium, isoferroplatinum and Pt-Rh-Ir-Fe-S-As mineralogy. Froth flotation showed that placer gold responded very well to all the collectors used, with gold recoveries of 82.7–99.8%. Flotation of platinum responded well to only potassium amyl xanthate, with a recovery of 80.4%. Results of low intensity magnetic separation showed that virtually all the liberated gold and platinum reported to the non magnetic product. A flowsheet, with estimated capital and operating costs, was developed to process 1500 tph of marine placer feed. Annualized costs per ton to process marine sediments were estimated to be $2.40 to $3.72 depending upon plant availability, 90% to 50%, respectively.

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Chapter 1 Introduction and Research Objectives

Recent price extremes of the six platinum group metals (PGM), which include iridium, platinum, palladium, osmium, rhodium and ruthenium, and the precious metal, gold, have created considerable interest in their production, uses and economics. Platinum was used for decorative purpose on a small scale as early as 1400 BC in Egypt[1]. The use of PGM as a catalyst in petroleum refining is well known, and their use as catalysts has evolved to include functions that reduce the emissions from automobiles. The catalytic properties of PGM are presently exerting a positive impact on the environment, through the development of automotive emission control catalysts. Platinum based catalysts are also suitable for eliminating volatile organic compounds, which are emitted from many industrial processes. PGM are used for a variety of tools and vessels to produce glass products such as lenses. Early uses included crucibles, wires in incandescent lights, thermometers and wires for telegraphs, in addition to decorative uses such as jewelry and silverware. Many other uses of PGM have been and are being developed.

The world-wide application of PGM has increased over the years and has reflected the growth in the number of applications of these technologically important metals. As the number of applications of PGM increase, the demand for more PGM and more efficient extraction technology is required. Early extraction of platinum consisted of gravity concentration followed by manual sorting of grains of natural platinum alloy. The gravity concentration methods which were used historically for platinum recovery, take

advantage of PGM high densities and low transportabilities in fluid systems (i.e. high settling velocities).

Onshore placer platinum group metals production of approximately 650,000 troy ounces (1926-present) has occurred near the concentrically zoned ultramafic complex at Red Mountain (Southwestern Alaska), along the course of the Salmon River and its tributaries[2]. According to Barker and Lamal[2] "It has long been suspected that placer PGM are concentrated in sediments offshore in the Bering Sea, west of Red Mountain. Page and others[3] suggest PGM resources of 5 million troy ounces contained in offshore deposits near Red Mountain and vicinity." Hindering the development of beach and offshore precious metals resources is the recovery of fine, flaky precious metals particles. Because of the very nature of precious minerals' densities, it can be appreciated that precious metals, which can be transported to and accumulate in marine environments, must be fine in size and/or flaky in shape[4,5]. For this reason, convectional gravity concentration methods are largely unsuccessful in recovering the fine, flaky precious metals found in marine environments. This, of course, is not true of some offshore deposits, which exist as submerged river or stream courses and which are an artifact of historic sea level changes. In such situations, the hydraulic conditions of the historic river or stream system controlled precious metals transportation and deposition.

Because of the shortcomings of gravity concentration recovery of PGM from a marine environment, froth flotation appears to have merits. This research combined other

mineral processing methods and froth flotation to concentrate the platinum and gold from a marine sediment sample from Southwestern Alaska. A flowsheet for recovering PGM and gold from these marine sediments was developed based on metallurgical test work from references and from this thesis research. An engineering economics analysis of the process flowsheet was conducted.

1.1 Research objectives

The objectives of this research were twofold. The first phase explored the possibility of using froth flotation to recover the precious metals found in marine sediments of Southwestern Alaska and specifically addressed

- flotation reagent selection
- gold and PGM recovery versus concentration ratio
- a kinetics study

The second phase utilized the results of the froth flotation research described above, previously determined gravity concentrator performance versus precious metals size and shape[6-9] and mineral processing engineering data to design a placer precious metals recovery flowsheet for a marine placer deposit located offshore of Red Mountain, Alaska. Accepted engineering design and cost estimation practices were then employed, in order to estimate both capital and operating costs to within 25% (conceptual cost estimate).

Chapter 2 Description of Goodnews Bay Platinum Deposit

The Goodnews Bay platinum deposits are located near the coast of the Bering Sea, near Red Mountain, Alaska. The deposits are 440 miles southwest of Anchorage[2] near the Yupik Eskimo village of Platinum, Alaska (Figure 2.1).

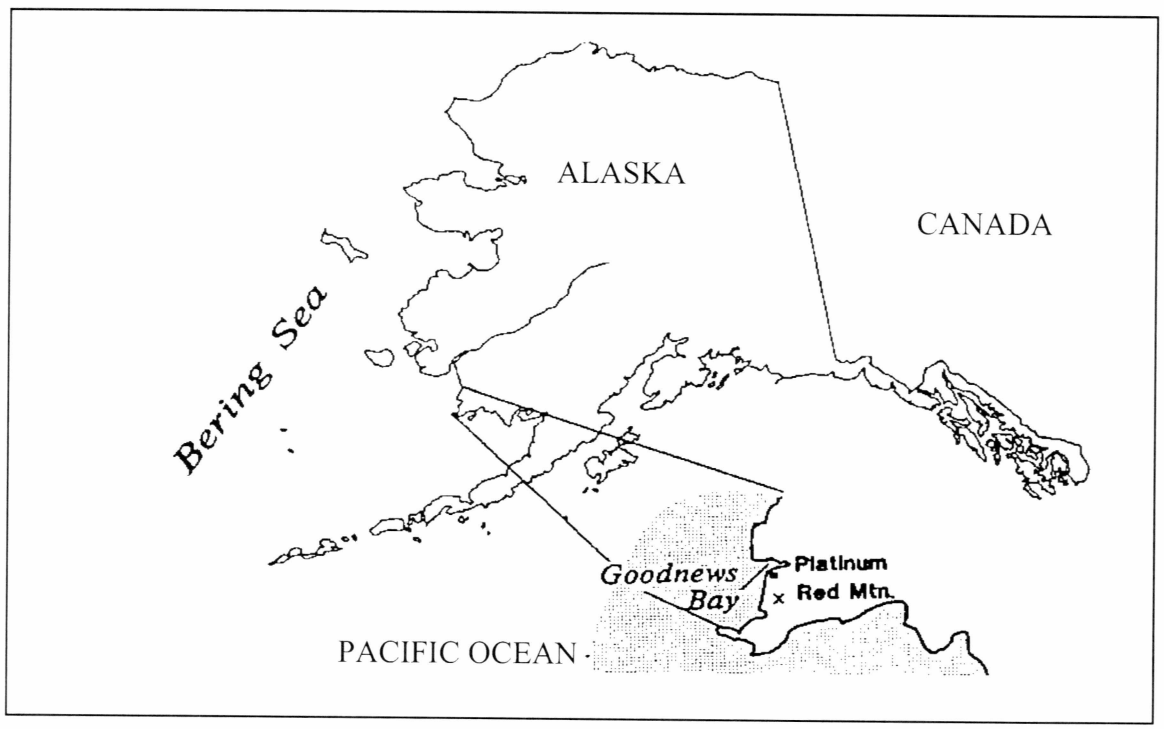


Figure 2.1 Map showing project area in Southwestern Alaska[2].

Alaska's history contains numerous episodes of precious metals production from beach and offshore deposits[10-13]. Among the 50 states, only Alaska can lay claim to being a commercial producer of placer platinum, the fabled metal of kings whose rarity exceeds that of gold. Gold production has always had a significant effect on the economy of

Alaska. Major gold production occurred in the offshore environment west of Nome, Alaska, where Westgold produced over 100,000 troy ounces of gold prior to ceasing operations in 1990[14].

2.1 Geology

A very informative survey on the geological setting of the ultramafic complex of the Goodnews Bay area can be found in Barker and Lamals' work[2]. The ultramafic complex forms an elongated body about 10 km in length that is of the Alaskan ultramafic intrusion type. The Goodnews Bay area lies in a region of subdued terrain, with landforms related to the Holocene continental glaciation[2]. In contrast to the low relief areas are Red Mountain, which rises 680 m above sea level, and Thorsen Mountain to the south, separating the Salmon River valley from the Bering Sea. Host rock in the area comprises greenschist-grade, metavolcanic and metasedimentary rock that were intruded by the concentrically zoned ultramafic complex that consists of dunite, peridotite and several varieties of clinopryroxenite[2].

Southworth and Foley[15] have revealed that the interpretation of gravity and magnetic data indicates that individual exposure is part of the larger south-dipping, convoluted, ultramafic sill-like mass, which is repeatedly exposed by one or more south-striking folds or faults. In common with other Ural-Alaskan type complexes, orthopyroxene is characteristically absent from ultramafic cumulates, a feature that has been taken to indicate an alkaline affinity[2].

A complex history of sea transgression and regression has affected the offshore geology near Red Mountain. According to Hopkins[16], much of the region of the Bering Sea was above sea level throughout the middle and late Tertiary. Sea regression coincides with glacial advance, intermittently exposing the broad coastal plain during the Pleistocene. With reference to Barker and Lamal[2], there is no known evidence of marine deposits in or above the glaciofluvial accumulations in the coastal bluffs near Red Mountains. Transgression[17] is still actively occurring, as evidenced by continuing encroachment of the surf against the bluffs.

2.2 Glaciation

Glaciers require very specific climactic conditions. Most are found in regions of high snowfall in winter and cool temperatures in summer. These conditions ensure that the snow that accumulates in the winter does not completely melt during the summer. Goodnews Bay is located in a transitional climactic zone, exhibiting characteristics of both a marine and continental climates. It has average precipitation of 22 inches of rainfall, with 43 inches of snowfall[17]. Red Mountain and its environs were glaciated by at least four glacial advances, dating in age from 900 years to greater than 45,000 years[17] before the present.

The extent of glacial scouring near or at the side of Red Mountain is an important factor regarding preservation of preglacial platinum group metals placers offshore[2]. Barker

and Lamal summarize the glaciation's effect near Red Mountain as only marginal glacial erosion with low-energy ice gouging; the principal ice contact was limited to the northern side of the mountainous mass[2].

2.3 Platinum deposits

Wagner[18] suggested that the first reference to the occurrence of platinum group metals associated with gold deposits was published in 1784 by Don Antonio de Ullo y Gracia de la Torre, of a Journey to Peru. PGM properties began to attract notice and aroused interest among scientists, which led to a series of ground-breaking investigations of platinum by many eminent scientists in the nineteenth century[18]. The high melting points of these metals limited their application, until scientists devised methods for consolidating and working platinum into useful forms.

The platinum group metals deposits may be classified into two main types: (1) endogenetic magmatic deposits which can be divided into (i) native platinum deposits (Uralian type) and (ii) complex copper-nickel sulfide deposit; and (2) exogenetic eluvial deposits[19]. The Uralian type is represented by deposits occurring in ultrabasic rock (dunite and more rarely periododites). The Goodnews Bay placer PGM deposits originate from dunite hosted, Uralian type rock sources. The platinum minerals, together with chromite, form small veined bodies, pockets, streaks and embedded particles.

Platiniferous placers are found in the Ural mountains, Colombia, Tasmania, and the United State (California, Alaska, and Oregon)[20]. Two types may be distinguished: (i) native platinum and native gold and (ii) placer enriched with osmium and iridium. With reference to Savitskii[19], the chief mineral types are isoferroplatinum, native platinum and native gold.

2.4 Placer mineralogy

The placer marine deposits of Goodnews Bay area contain appreciable amounts of native gold, platinum group metals and the accessory minerals; magnetite, ilmentite, chromite, zircon and cinnabar[2]. Barker and Lamal[2] identified and isolated numerous particles of isoferroplatinum (50-500 microns), osmiridium (50-100 microns) and gold (300-500 microns) during their 1981-1988 marine sampling programs. Also, mineralogical investigation using the binocular microscope in Mineral Industry Research Laboratory, University of Alaska Fairbanks, has proved that the beach deposit contain appreciable amount of gold and PGM. Platinum group metals found in the Goodnews Bay complex and the environs of Red Mountain display a chemical affinity for chromite and magnetite[2]. Barker and Lamal[2] also showed that the offshore concentrate contained PGM which are classified into two main groups, isoferroplatinum and osmiridium and three minor classifications (Table 2.1), as well as native gold. Native gold was coarser and more abundant in the studies of Barker and Lamal than the PGM.

Table 2.1 Placer mineralogy of Goodnews Bay, Alaska [2].

Minerals	Composition	Characteristics
Major Minerals		
Isoferroplatinum	$(Pt, Pd)_3(Fe,Cu)$	68% to 90% by weight Pt and 1% to 5% Ir, Os, Rh; weakly magnetic grains up to 50-500 μm.
Osmiridium	Ir, Os	50% to 80% Ir, 6% to 30% Os, minor Pt, Ru, and Fe; typically 50 to 100 μm.
Gold	Au	Grains are in the range of 300 to 500 μm. Ag was only detected in solid solution alloy.
Minor Minerals		
Platiniridium	$Ir_{58-84}Pt_{10-32}Fe_{6-10}$	Only 2 grains observed.
Sperrylite	$Pt As_2$	Generally well-rounded grains interlocked with moncheite.
Moncheite	$(Pt, Pd) (Te,Bi)_2$	Interlocked with sperrylite.
Accessory Minerals	Magnetite, ilmenite, chromite, zircon, cinnabar.	

Chapter 3 Literature Review

3.1 Characteristics of precious metals

The best known precious metals are gold and silver. Platinum group metals are another group of precious metals of which platinum is the most commonly known. Chemically, the precious metals, are less reactive than most other metals, and have high luster and melting points. Historically, silver and gold have been important as currency and jewelry. PGM have been primarily consumed for industrial applications.

Gold is a soft, shiny, yellow, dense, malleable, ductile metal. Gold, which occurs in eighteen isotopes, is softer than most of the PGM and silver. The specific gravity of pure gold is 19.3. Its hardness according to Rose[21] is 2.5 to 3. Gold has a melting point of 1063°C. Pure gold is a good conductor of heat and electricity. Heat, moisture, oxygen, and corrosive agents have very little chemical effect on gold, making it well-suited for use in jewelry[15] and as a conductor in micro processors.

The six platinum group metals fall into two groups[18]:

- the light platinum group metals, which consist of ruthenium, palladium, and rhodium, with specific gravities ranging from 12.1 to 12.4, and
- the heavy platinum group metals, consisting of platinum, osmium and iridium, with specific gravities ranging from 21.5 to 22.5.

Platinum is by far the most important and widely used member of the groups. Platinum is a lustrous greyish-white, malleable, ductile metal and a member of group 10 of the

periodic table of the elements. It has the third highest density, behind osmium and iridium. Platinum is chemically unaffected by air and water, but will dissolve in hot aqua regia[18], and also in hot concentrated phosphoric and sulfuric acids, and in molten alkali.

Palladium, which is a light platinum metal, ranks next to platinum in importance. Palladium is a white metal and has a color between platinum and silver. It is malleable and ductile and dense and has the lowest melting point of the platinum group metals[18]. Palladium is chemically attacked by sulfuric acid, but dissolves slowly in hydrochloric acid, and it is the least dense of the platinum group metals. Palladium does not chemically react with air at normal temperatures.

With reference to Wagner and others[18,19], iridium is white, resembling platinum with a slight yellowish cast. It is very hard and brittle and is not chemically attacked by either acid or aqua regia. Iridium is the most inert of the precious metals. Its chemical resistance is legend, and is superior in that respect to gold and platinum, while featuring a far higher melting point. As a result, it is used for making crucibles and other apparatus for use at high temperatures. It is also used in certain electrical contacts and also as a hardening agent for platinum.

Ruthenium is a hard grayish-white metal. Ruthenium dissolves in fused alkali, but it is not attacked by acid[19] and is normally used in hardening other PGM.

Osmium and rhodium are respectively classified by Wagner[18] as heavy and light platinum group metals. Osmium is a hard, brittle, blue-gray or blue-black transition metal. Osmium is the densest natural element.

Rhodium is a hard silvery white and durable metal that has a high reflectance[18]. Rhodium has both a higher melting point (Table 3.1) and lower density than platinum[18]. Rhodium is not chemically attacked by acid, but it dissolves in aqua regia.

Table 3.1 Physical and chemical properties of PGM[22].

	Light PGM			**Heavy PGM**		
	Palladium	Rhodium	Ruthenium	Platinum	Iridium	Osmium
Symbol	Pd	Rh	Ru	Pt	Ir	Os
Specific gravity	12.16	12.44	12.10	21.45	22.42	22.47
Hardness	4.8	-	6.5	4.3	6.5	7.0
Melting point	1555^0C 2830^0F	1964^0C 3567^0F	2334^0C 4233^0F	1768^0C 3214^0F	2446^0C 4435^0F	3033^0C 5491^0F
Valency	1, 2, 3	2, 3, 4	2, 3, 4, 8	2, 4	2, 3, 4	2, 3, 4, 8

3.2 Froth flotation

Valuable minerals in an ore can be separated from worthless gangue minerals by the froth flotation process[23]. This process was developed in Australia at the start of the 20th century to treat the sulfide silver/lead/zinc ore at Broken Hill, NSW. Since then, froth flotation has become a key unit process in the recovery of most of the world's copper, lead, molybdenum, nickel, platinum group metals (sulfide ores), silver, and zinc, and in the treatment of certain gold and tin ores. Froth Flotation is used extensively in industrial applications for the treatment of finely disseminated ores.

Froth flotation is a surface-chemistry based process that takes advantage of the differences in wettability at solid particle surfaces. Solid surfaces are often naturally wettable by water and termed hydrophilic[23]. A surface that is non-wettable is water repelling and termed hydrophobic. If a surface is hydrophobic, it is also typically air attracting, termed aerophilic, and is strongly attracted to an air interface, which readily displaces water at the solid's surface. Using froth flotation, the separation of a mixture of solids in a slurry may be accomplished by the selective attachment of hydrophobic solid particles to gas bubbles (typically air) injected into the slurry.

Froth flotation is often used to separate solids of similar densities and sizes, which prevent other types of separations based upon gravity concentration. It is especially useful for particle sizes below 106 μm (150 mesh)[23]. The lower size limit for conventional flotation separation is approximately 38 μm (400 mesh); however, particles

as small as 38 μm have been separated[23]. At these small particle sizes, it may be difficult to take advantage of surface-property differences to induce selective hydrophobicity. On the other hand, particles greater than 200 μm (65 mesh) tend to be readily sheared from the bubble surfaces by collision with other particles or vessel walls[23].

3.3 Particle surface properties

Surface is a two dimensional unit length and breadth lacking thickness. It is usual to regard the changes which take place on the surface of the mineral particle in pulp as resulting from an equilibrium state between hydrated ions (complexes) in solution and the charged surface of the particle. The mineral ions will migrate into the aqueous phase, if the polar strength of the water molecule is greater than the latice attraction binding mineral ions[24]. A particle which has been sheered naturally by a crusher, mill or by natural degradation has exposed its surface to many partial faces of unit form[24], the atoms of this cell are arranged according to a definite pattern and this results in the ability of the nucleus to give or receive electrons, resulting in different bonds. Water is the usual medium in which the separation of mineral occurs by flotation. Water, a dipole compound, contributes to the main process of bubble adhesion, the properties of surface minerals and the properties of surface active reagents.

3.4 Wettability

Froth flotation involves three phases; solid, liquid, and gas and the corresponding potential phase interfaces; solid-liquid, solid-gas, liquid-gas, and solid-liquid-gas. The hydrophobic (aerophilic) or hydrophilic nature of the solids at the solid-liquid-gas interfacial region is determined by the wettability of the solid. A parameter that characterizes wettability is the contact angle at the three-phase interface. The development of the contact angle between the mineral surface and the bubble surface is due to the force which tends to separate particle and bubble as shown in Figure 3.1[21].

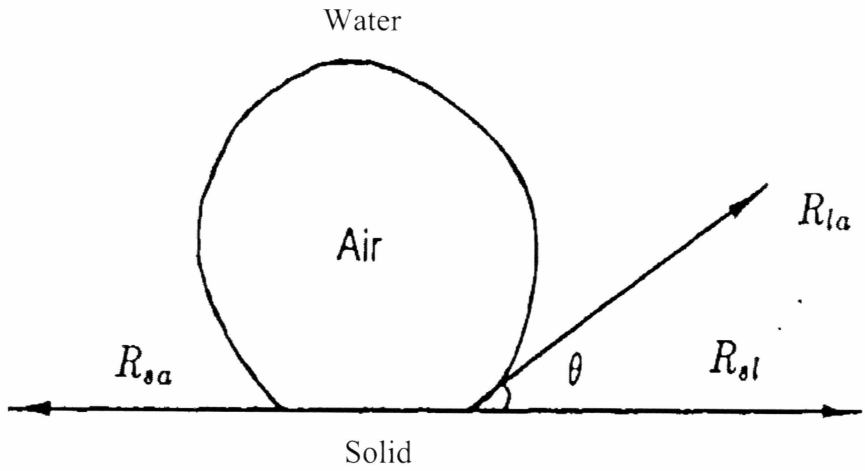


Figure 3.1 Contact angle between bubble and particle surface in aqueous medium[21].

The contact angle is therefore used as an investigative tool for understanding the flotation response[23]. The contact angle, θ is related to the respective interfacial surface tensions, R_{sa}, R_{sl} *and* R_{la}, using Young's equilibrium equation,

$$R_{sa} = R_{sl} + R_{la}\cos\theta \qquad [3.1]$$

The force required to break the particle-bubble interface is called the work of adhesion (W_{sa}), and equals the work required to separate the solid-air interface and produce an air-water interface and a soild–water interface. That is

$$W_{sa} = R_{sl} + R_{la} - R_{sa} \qquad [3.2]$$

$$W_{sa} = R_{la}(1-\cos\theta) \qquad [3.3]$$

Equation 3.3 results from combining equations 3.1 and 3.2. With reference to equation 3.3, it can be seen that the greater the contact angle, the greater the work of adhesion between particle-bubble interface and the more resilient the system is to disruptive force. Contact angles in flotation systems typically do not exceed 100°. In flotation plant practice, the system is operated in such a manner that one of the solid components has a greater hydrophobicity than the other solids from which it is to be separated. Generally this is achieved by chemical addition, in order to affect mineral surface hydrophobicity.

3.5 Electrical double layer

The reactions between water molecules and various mineral surfaces are not all equal; the free energies of the solution vary, and some minerals or metals will pass into solution with greater ease than others. Accordingly, the electrical balance of the particle surface becomes charged, since the mineral surface acquires charge, which is opposite in sign to the ions which have been dissolved. The electrical double layer is therefore the

separation of charges at the boundary, which occur due to the increased concentration of one type of ion at the surface, which hinders the migration of other types of ions into the solution[24]. The ions found in the vicinity of the minerals are strongly bonded to the surface, as the mineral particle moves, they move with it. The ions found farther from the mineral surface form a layer often referred to as a diffuse layer (Figure 3.2 Row 5). Figure 3.2 shows a schematic representation of the electrical double layer existing at a mineral surface. At a point, the diffuse layer is sheared from the surface during movement, the electrical equilibrium of the system is disrupted and a potential difference is generated between the liquid and the moving particle[24]. This voltage or potential difference is termed the zeta potential or electro-kinetic potential.

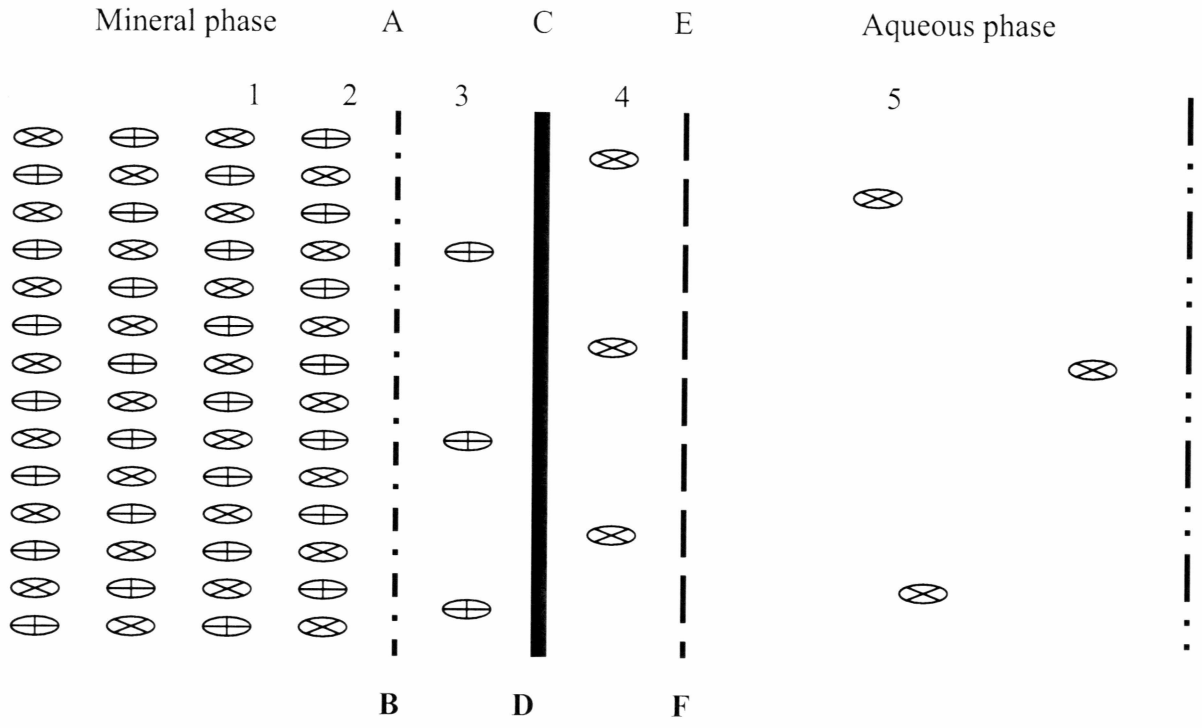


Figure 3.2 Schematic illustration of the electrical double layer[24].

⊗ -positively charged species

⊕ -negatively charged species

AB -represents the undisturbed lattice of the mineral

CD -represents the physical boundary of the mineral

EF -represents the boundary of the liquid, which is strongly bonded to and moves with the particle.

Rows 1 and 2 -undisturbed ions in mineral

Row 5 -the diffused ion layer

3.6 Mineralization of air bubbles in flotation

The basis of the froth flotation process is mineral particles attachment to air bubbles. All conditions such as sizing, treatment with reagents, and pulp agitation in the flotation cells are for the purpose of producing the correct conditions for rapid, selective and strong attachment of bubbles to minerals[23]. Bubble attachment in flotation is normally achieved by one of the following methods: (i) a particle coming in contact with a bubble already formed in the pulp, or (ii) a gas bubble being precipitated from solution onto the particle surface.

3.6.1 Separation kinetics

Separation kinetics is used to establish the nature, condition and speed of a froth flotation of minerals. The dynamics and kinetics of froth flotation separation involves the surface phenomena of the collector's chemical envelope surrounding the solid particles and its hydrophobic nature[25]. With reference to Perry and Green[25], particle size and the mass of material has an effect on the ability of the bubble to successful transport the solid particle to the mineral that lies beneath the hydrophobic envelop. Conditioning time is another parameter which affects the flotation separation. If enough conditioning time is given for collector action to selectively render the desired solid particles hydrophobic, the remaining overall separation is essentially dependent only upon the efficiency of the bubble and solid particle interactions. The overall separation kinetics[25] of flotation is expressed as

$$r_s = \left[\frac{dC}{dt}\right] = -kC^n \qquad [3.4]$$

where C is the concentration of the desired component in the flotation cell; t is the time; n is the order of flotation process and k is the flotation rate constant.

In flotation kinetics, it is often to assume a reaction order and subsequently determines the accuracy of that assumption based upon the experimental data[22,26]. Kinetic separation assuming a first-order process permits one to at least compare systems[26]. This is often termed a pseudo-first-order assumption. Solution of this equation assuming first-order kinetics, $n = 1$, yields the integrated rate expression:

$$ln\,C = -kt + I \qquad [3.5]$$

$$\left[\frac{C}{C_o}\right] = e^{-kt} \qquad [3.6]$$

where C_t is the concentration of the desired component (the hydrophobic component) in the flotation cell at time t and C_o is the concentration of the desired component at $t = 0$, at the beginning of the separation. A classical first order rate equation which has been proposed by majority of researchers[27,28] in flotation kinetics studies is represented as follows:

$$R = R_\infty[1 - \exp(-kt)] \qquad [3.7]$$

where R is the percentage recovery of the mineral at any time t, R_∞ is the maximum percentage recovery of the mineral at infinite time t_∞ and k is the rate constant (s^{-1}).

3.6.2 Phase

The froth flotation process involves three major phases interacting at the same time: solid (mineral), liquid (aqueous solution) and gas (air). The pulp phase is determined in terms of solid-liquid ratio. There, when the mineral has gained enough hydrophobicity[29,30] achieved often by collector adsorption, the solid particles are ready to interact with the rising front of bubbles. The universal liquid for froth flotation is water, though in a few instances saturated brine[31] and seawater[32] were used. Various reagents are added to the liquid phase for selective control of the desired frothing. Air is the common gas phase used in flotation. The solid phase varies considerable. It basically depends on the valuable mineral in the ore body.

3.6.3 Flotation efficiency

Flotation efficiency[23,25], which is defined by equation 3.8, depends on mechanically satisfactory froth. This depends on frothing mechanism of the frother combination for a particular flotation application. In majority cases, a frother containing at least a portion that is an alcohol such as methyl isobutyl carbinol, 4-methyl-2-pentanol (MIBC) is the best choice to match most collectors. Flotation efficiency (E_f) is represented as follows:

$$E_f = \frac{3Gkh}{2d_bV} \qquad [3.8]$$

where G is the air flow rate; d_b is the average bubble diameter; V and h are the volume and depth of the particle suspension, respectively, and k is the flotation rate constant.

3.6.3.1 Dynamic froth stability

Froth breaks down through rupture due to the film thinning as shown by Ewer and Sutherland[33], who states that, the key factor for froth stability is the movement of inter-bubble liquid away from the point of potential rupture: that is, the surface liquid moves from a region of low surface tension. They went further to conclude that, the film will be stable when the surface tension is highest at the centre of disturbance.

3.7 Flotation reagents

The surface chemical reagents used for the process of froth flotation can be classified broadly either as collectors, activators, depressants, dispersants or frothers. The selection of reagents is usually dependent on the ore characteristics, the concentration of various or constituents' mineral, the target of concentrate grade and recovery[34]. Addition of flotation reagents and introduction of air bubbles in flotation of mineral particle are meant to produce the following effect: - (i) to form froth, (ii) to stabilize the froth, (iii) to coat the mineral particle with an ultra-microscopical layer of oil or equivalent chemical, (iv) to cause attachment of air bubbles to the oiled particle thus assisting them to rise, (v) to assisting in intensifying the froth formation and (vi) to assist in intensifying the gangue depression[21].

3.7.1 Collector

To invoke selective hydrophobicity, substances know as *collectors,* also called *promoters* are used. Collectors are typically heteropolar organic substances – they contain both non-polar and polar chemical groups. The non-polar end is almost always a long-chain or cyclic hydrocarbon group that is hydrophobic. The collector must be able to attach to the solid and it does so through its polar end, which is typically an ionic group termed at the solidophil group[23]. To a lesser extend, non-ironic collector are also used.

During flotation experiment, the non-polar end of the collector orients outward from the solid surface forming a non-polar chemical envelope surrounding the solid particle, inducing hydrophobic behavior on the solid surface. The solid particle can then, more readily attach to an air bubble. Collectors are classified according to ability to dissociate into ions in an aqueous solution and with regard to the type of ion which produce the water repellent effect[24].

3.7.1.1 Choice of collector

The principal factors in the selection of a collector for a flotation process are the mineral forms (sulfide, oxide, non-sulfide or metallic species), complexity of the ore (the association of valuable minerals with other and gangue minerals) and dosage of the collector[35]. For industrial sulfide mineral flotation, sulphydryl collectors such as xanthate, dithiophosphate, dixanthogen and thiocarbabilide are used[36]. The solubility, pH application range, physical state and adsorption characteristics vary widely for

different compound with typical dosage ranges from 0.01 to 0.1 kg/ton of dry feed to flotation[36] and with some application ranges up to 0.5 kg/ton[34].

For non-sulfide mineral flotation, the hydrocarbon chain length of the collectors ranges from 12 carbon atoms to 22 carbon atoms. The most effective collectors for the inherently naturally floating mineral are those of uncharged and water insoluble collectors[35]. Such collector operates best at or near natural pH of mineral slurry[37]. Example of uncharged and water insoluble collector includes: thionocarbamates ($R'HN-(C=S)-OR''$), mercaptans, and aqueous aerofloat promoters xanthogen formats ($R'O-(C=S)-S-(C=S)-OR''$). Klimpel[36] gave a number of reasons why insoluble collectors are preferred for the recovery of inherent natural floating species. This includes:

- Use of water soluble collector such as xanthates and dithiophosphate slow down the rate of flotation.
- High dosage of water soluble collectors is required for equivalent optimum equilibrium recoveries than required for water insoluble collector.
- Use of uncharged water soluble collector helps natural floatable mineral to be naturally selective over many gangue mineral including pyrite. The use of charge water soluble collector destroys this selectivity.

3.7.1.2 Interaction of thiols with metals

Wood and Gardner[38] proposed that "interaction of thiols on metal (platinum and gold) involves an electrochemical mechanism, with anodic oxidation of the collector supported a cathodic (reduction) reaction (e.g. reduction of oxygen)":

$$2X^- = X_2 + 2e^- \qquad [3.9]$$

$$\tfrac{1}{2}O_{2(ads)} + H_2O + 2e^- = 2OH^- \qquad [3.10]$$

In these equation, X^- indicated the xanthates ion and X_2 dixanthogen. Wood and Gardner[38] further proposed that equation 3.9 proceeds via an initial chemisorption step and that the multi-layer of the dixanthogen product are bound to the chemisorbed monolayer xanthates by the reaction of the hydrocarbon parts of the molecule. The main experimental advantages of the electrochemical nature of these reactions can be tracked by the electrochemical techniques.

3.7.2 Frother

A second class of chemical agent is the floating agent. Frothers are used to provide stable froth, increase the rate of recovery[36], and to control the rate of coalescence of the individual bubble in the flotation cell. Frothers are generally classified by their polar groups with the most common being the hydroxyl (–OH), carboxyl (–COOH), carbonyl (=C=O), amino (–NH3), and sulpho (– OSO2OH and–SO2OH) groups. Effective frothers typically contain at least five or six carbon atoms in their straight-chain and non-polar group to obtain sufficient and stable interaction with the air phase[23]. For

branched-chain hydrocarbons, the number of carbon atoms in the non-polar group may range up to sixteen. Branched-chain frother such as pine oil, MIBC (methyl isobutyl carbinol, 4-methyl-2-pentanol), glycol ($H-(OC_3H_6)_n-OH$) and polypropylene glycol methyl ethers (PGME) are very effective for free gold flotation and precious metal from the recovery point of view[33].

3.7.3 Modifying agent

Modifying agent can be grouped into the following categories: pH regulators, activators, depressant, dispersants, and flocculants[36]. The major pH regulators used today in most industry include lime, soda ash, caustic soda, and hydrochloric acid. With reference to Klimpel[35], lime which is frequently used as a pH regulator has a negative effect on gold flotation. In other system the influence of pH can be complicated causing simultaneous changes in many of the variables of flotation[36].

Activators are modifying agent, when introduce in the flotation process cause better collector attachment on the desired mineral surfaces. Another important mechanism of activator is the adsorption of hydroxyl complex of a metal onto a component surface near the pH precipitation of that metal[36]. Cupper sulfate is often used as activator in promoting free gold flotation[35].

Depressants prevent floating of undesired component that without special treatment will have the tendency to float. Slime, for example, prevents collector attachment by

adsorbing the collector, decreasing its availability to the mineral species to be floated. Sodium meta silicate (Na_2SiO_3) in solution increases the total charge on the particle, so that the slime layer readily disperses and the clean mineral surface can then interact with the collector[39]. A common depressant mechanism applied in the industry is deactivation where care is taken to remove situation that normally leads to undesired activation of some component[36].

3.8 Gold flotation

Gold ores are commonly classified into two major categories: free-milling and refractory ores. Generally, placers, quartz vein gold ores and oxidized ores are free-milling. Gold in these ores can be recovered by gravity concentration[40]. Depending on the size of gold present in the deposits, flotation may be used in the gold recovery flowsheet, as may cyanidation. Klimpel[35] provides an excellent review of gold flotation and makes specific collector and frother recommendations to improve recovery based on his experience with industrial scale flotation plants. American Cyanamid classified gold flotation into the following classes[41]:

1. Gold ores in which only free gold is present. This class of ore usually has a low sulfide content.
2. Gold ores in which a portion of the gold is free, either in metallic form or as telluride, the balancing being associated with sulfides such as pyrite.
3. Ores, whose major values are base metals, such as cupper, zinc and lead, and gold is of secondary value.

American Cyanamid research has revealed that, in the gold flotation of classes 1 and 2, Aerofloat 208 and Aerofloat 3477 dithiophosphate promoters and xanthate promotors are widely used. Flotation of classes 1 and 2 are usually carried at natural pH. A strong alcohol frother type may be used, due to the free gold particles present. Aero 404 or 407 mercaptobenzothiazole, as a principal collector, and xanthate as secondary collector, can also be used, where more powerful collectors are required[41].

Wang and Poling[5] provide an excellent historical and modern review of placer gold flotation research. Various flotation studies, reviewed by Wang and Poling, have shown successful placer gold flotation results with commercial xanthate-dithiophosphate collectors and frothers. Similarly, Cook and Rao[4] used American Cyanamid reagents Aero 301 (sodium secondary butyl xanthate), Aerofloat 208 (sodium diethyl and sodium butyl dithiophosphate mixture) and Aerofloat 15 (aryl dithiophosphate acid) as collectors to successfully float marine placer gold from Cape Yakataga, Alaska in laboratory tests; gold recoveries ranged from 86% to 100%.

3.9 PGM flotation

Research has established that the highest recoveries of sulfide hosted platinum group metals are achieved with a combination of high strength xanthate, such as amyl or isobutyl xanthate, and Aerofloat 3477 (sodium isobutyl dithiophosphate) or Aerofloat 5430 (dithiophosphate)[41]. Research on flotation behavior of placer platinum group alloy mineralogy is very limited. Volyanskii[42] investigated the reaction of two collectors,

potassium butyl xanthate and sodium dibutyl xanthate with sperrylite [$PtAs_3$]. The significance of his investigation was that the collectors are consistent with the formation of dixanthogen on the surface of the sperrylite and the research also mentions the formation of platinum xanthate species, resulting from interaction of sperrylite with the collectors. According to Vermaak[43], Anglo Platinum Research Centre (ARC) has assigned some physical characteristics and the recovery characteristics to PGM, based upon mineralogy and the ore type. Table 3.2 gives a general idea of how amenable different ores of PGM are with respect to froth flotation and gravity concentration[43].

Table 3.2 Physical and recovery characteristics of different ores of the PGM[43].

Class	Size	Density	Gravity Recovery	Flotation Recovery
Sulfides	Often Coarse	High	Good	Good
Telluride	Variable	Low	Poor	Poor
Arsenide	Variable	Medium	Good	Good
Alloys	Often Coarse	Very high	Excellent	Variable
Oxides	Fine	Low	Very poor	Very poor

With reference to Glembotski[44]:

> "Platinum-bearing ores may be divided into dunites, in which the platinum grains are attached principally to chromite schlieren, and sulfide ores,

> principally cupper-nickel ores. Native metals are found in the dunites ore, while native metals, sulfide and platinum arsenide (sperrylite) are found in the sulfide ores. Flotation is used principally for concentration of sulfide platinum ores. In most cases, platinum-bearing minerals can be readily extracted by flotation. The native minerals (ferroplatinum and cuproplatinum), the composition of which include iron and copper, give rise to the greatest difficulty. When the later become oxidized, a film of oxides, hydrates and carbonates of iron and cupper forms on the grains surfaces."

Contact with the metallurgical staff at the Stillwater Mine in Montana has revealed that isoferroplatinum exists in the ore body; up to 10% of the platinum may exist as isoferroplatinum compounds [45]. The basic reagent suite for processing the complex ore body consists of potassium amyl xanthate, a strong dithiophosphate, carboxyl methyl cellulose as a talc/serpentine depressant and some minor amounts of MIBC[45].

Chapter 4 Sample Collection and Preparation

The samples collected for this study were taken at GPS coordinates, N58° 52.526', W161° 46.761', approximately 1.5 miles north of the mouth of the Salmon River. Samples were collected in a manner to insure that they contained considerable, visible placer gold and platinum; i.e. they were high grade samples. As such they should not be considered representative of any deposit. High grade samples were required for this metallurgical test work, in order to allow for accurate metal balances, given the free particulate nature of the gold and PGM in these marine sediments. These samples were used for microscopic studies, magnetic separation studies, a particle size analysis, and froth flotation tests. This chapter deals with the method of sample collection and sample preparation.

During the summer of 2005, Professor D.E. Walsh and Professor H.K. Lin of the Mineral Industry Research Laboratory (MIRL), University of Alaska Fairbanks collected samples of beach sand for the project. The field samples (Figure 4.1 and Figure. 4.2) were collected from visible black sand layers and from black sand enrichments on a ferricrete layer. Sample acquisition was greatly facilitated by the previous work and excellent reporting of Barker and Lamal[2], and by the logistical support provided by Mark Moyle, resident of Platinum, Alaska. Field beach sand samples were collected, per the protocol shown in Figure 4.3, producing six buckets of sample; (i) 4, five gallons buckets of –6 mesh pan feed material, and (ii) 2, five gallon buckets of –6 mesh pan concentrate. Panning reduce the volume of pan feed material by approximately 70%.

Figure 4.1 Sample collection site south of Red Mountain.

Figure 4.2 Black sand enrichment layer at sample collection site.

The six buckets of material weighing approximately 136 kg (300 lb) were shipped to MIRL's mineral processing laboratory for metallurgical studies. In the course of this thesis research, only one buckets of the pan concentrate sample was consumed. The remainder of the sample remains for future studies at MIRL.

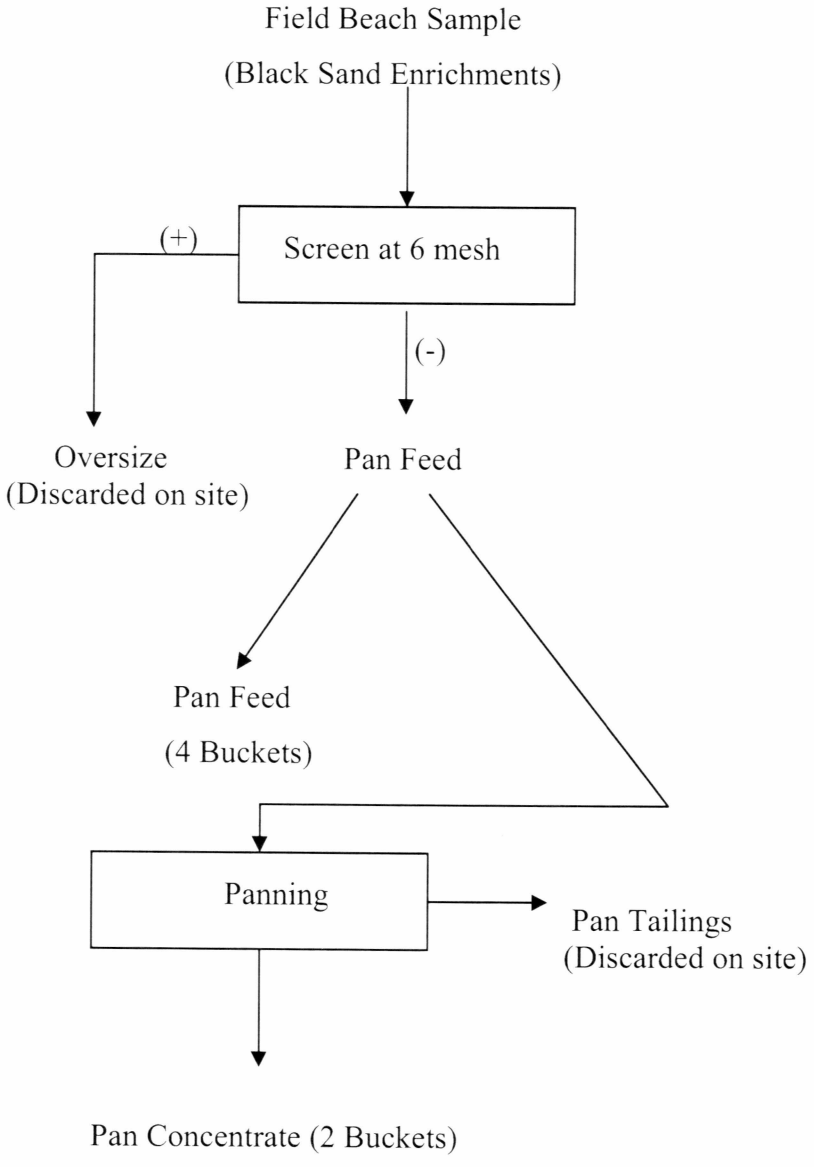


Figure 4.3 Field sample collection and processing.

Chapter 5 Description of Metallurgical Test Work

One bucket of pan concentrate sample was wet screened at 20 mesh, then both +20 mesh and the –20 mesh fractions were dried and weighed. The +20 mesh material was only 1.6% of the pan concentrate by weight.

A rotary splitter (manufactured by Prosplitter Desert Lab) at the Fort Knox Mine (Fairbanks, Alaska) was used to split the dry –20 mesh pan concentrate into 20 representative samples. Each split was placed into a plastic sample bag and weighed (Table 5.1). Masses of the splits ranged from 960 grams to 1110 grams.

5.1 Low intensity magnetic separation

A low intensity magnetic separation was carried out to estimate the magnetite content of the pan concentrate and the distribution of platinum and gold between the magnetic and non magnetic products. A Carpco, dry, low intensity, magnetic separator was used to conduct a straightforward magnetic separation experiment on a split of –20 mesh pan concentrate. After thoroughly cleaning the low intensity magnetic separator, the gap (clearance) was adjusted to 12 mm using a metric rule. The sample was then run through the magnetic separator at a feed setting of 45-50% with a rotor speed of 50 rpm. The time to process the sample was recorded. With the clearance re-adjusted to 6 mm, the non magnetic product from the first separation stage was re-run through the low intensity magnetic separator at the same feed setting and rotor speed. With a clearance re-adjusted again to 3 mm, the non magnetic material from the second stage was re-run through the

low intensity, magnetic separator at the same settings of feed rate and rotor speed. Figure 5.1 shows the protocol for the magnetic separation test. Chapter 6 contains the results of the magnetic separation test.

Table 5.1 Mass of representative pan concentrate splits.

Split No.	Mass [g]
1	1,088
2	1,056
3	1,059
4	1,093
5	1,089
6	1,063
7	1,065
8	1,077
9	1,005
10	1,069
11	1,066
12	960
13	1,053
14	1,082
15	1,056
16	1,110
17	1,038
18	1,071
19	1,070
20	1,074

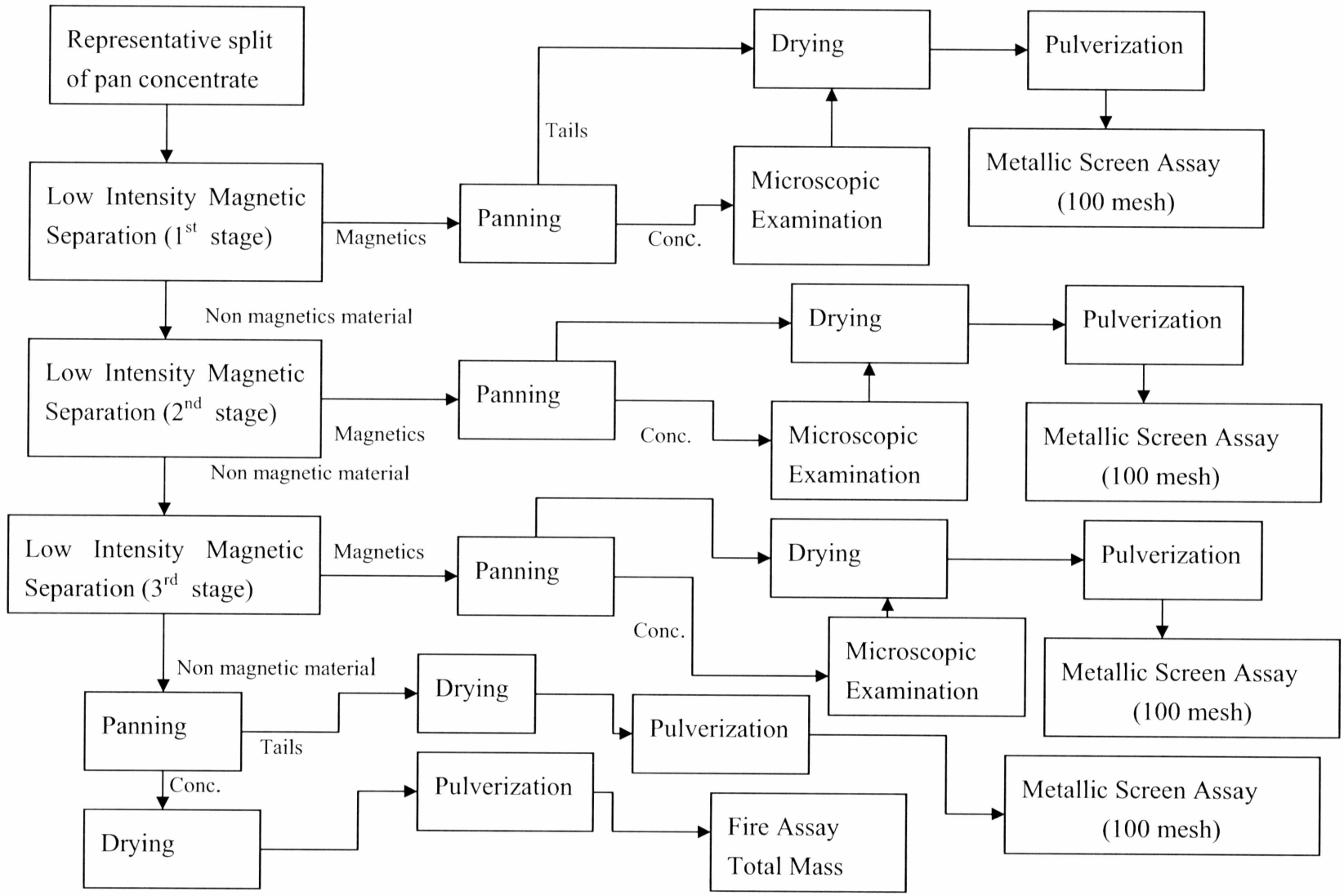


Figure 5.1 Flowsheet of the low intensity, magnetic separation test procedure.

5.2 Particle size analysis

Though this sample was not representative of future, mineable material, for completeness a representative split (1,069 grams) of the pan concentrate was used for particle size analysis. ASTM sieves and a Ro-Tap sieve shaker were employed. The particle size distribution of the -20 mesh pan concentrate and the assay results for individual size fractions are presented in Chapter 6.

5.3 Flotation experiment

Based upon the placer gold flotation research by Wang and Poling[5] and Cook and Rao[4], and the recommendations for liberated hardrock, gold flotation by Klimpel[35], laboratory flotation experiments were designed, in order to screen promising, commercially available, flotation collectors for their effectiveness in recovering placer gold and placer PGM from the pan concentrate sample. With the assistance and cooperation of Cytec Industries Inc., five collectors were obtained for testing:

- Aero 350 (potassium amyl xanthate)
- S-8474 Reagent (diakyl dithiocarbamate)
- Aero 3477 (sodium isobutyl dithiophosphate)
- Aero 3320 (xanthate allyl ester)
- Aero 5100 (allyl alkyl thionocarbamate)

A dosage of 100 mg collector per kilogram placer pan concentrate was selected for the flotation screening. Three frothers were chosen based on the research of Cook and Rao[4] and Klimpel[35]:

- Pine oil
- MIBC
- Aero 65 (polypropylene glycol)

Based on the result of particle size analysis and size fractions assays, representative splits of the pan concentrate were screened at 40 mesh. The undersize (-40 mesh) was used for flotation tests. A 2-L Denver flotation cell was employed for flotation test work. Cell volume was chosen to yield a 27-30% solid by weight pulp density. The laboratory flotation experiments followed the protocol outlined below:

1) Solid sample and water were added to the 2 liter flotation cell.
2) Agitation was initiated at 1300 rpm.
3) An attrition scrubbing period of 5 minutes was allowed.
4) Collectors (Table 5.2) were added to the cell.
5) A conditioning period of ten minutes was allowed.
6) Frothers (Table 5.3) were introduced to the cell.
7) A conditioning period of five minutes was allowed.
8) A baffle was inserted onto the flotation cell and pressurized air flow was introduced at a rate of 6-6.5 l/min.
9) Froth was allowed to build up for two minutes.

10) Froth was skimmed off for two minutes.

11) Froth was allowed to build up for two minutes.

12) Froth was skimmed of two minutes.

13) Air flow was switched off and frothers were added for the second stage flotation.

14) A conditioning of five minutes was allowed.

15) Air flow was re-introduced at a rate of 6-6.5 l/min.

16) Froth was allowed to build up for two minutes.

17) Froth was skimmed off for two minutes.

18) Froth was allowed to build up for two minutes.

19) Froth was skimmed off for two minutes.

20) Air and agitation were shut off and concentrate and tailings samples were prepared.

Table 5.2 Collector dosages used in flotation tests.

Flotation test	Mass of – 40 mesh Feed (g)	Collector	Collector Dosage (mg/kg)	Pulp Density
Flotation 'C' Test	796.20	Aero 350 Xanthate **($C_6H_{11}OS_2K$)** Xanthate Salt	100	27%
Flotation 'D' Test	850.80	S-8474 Reagent **($C_9H_{18}NS_2Na$)** Dithiocarbamate	100	30%
Flotation 'E' Test	834.20	Aero 3477 **($C_8H_{18}O_2PS_2Na$)** Dithiophosphate	100	29%
Flotation 'F' Test	803.9	Aero 3320 Xanthate ester	100	28%
Flotation 'G' Test	861.8	Aero 5100 Thionocarbamate	100	29%
Flotation 'H' Test	885.1	Aero 3477 **($C_8H_{18}O_2PS_2Na$)** Dithiophosphate	100	29%
Flotation 'I' Test	880.4	Aero 350 Xanthate **($C_6H_{11}OS_2K$)** Xanthate SALT + Aero 3477 **($C_8H_{18}O_2PS_2Na$)** Dithiophosphate	50 50	30%

Table 5.2 (continued) Collector dosages used in flotation tests.

Flotation test	Mass of – 40 mesh Feed (g)	Collector	Collector Dosage (mg/kg)	Pulp Density
Flotation 'K' Test	836.9	Aero 350 Xanthate **($C_6H_{11}OS_2K$)** Xanthate SALT + Aero 3477 **($C_8H_{18}O_2PS_2Na$)** Dithiophosphate	50 50	29%
Flotation 'L' Test	835.7	Aero 350 Xanthate **($C_6H_{11}OS_2K$)** + S-8474 Reagent Dithiocarbamate + Aero 3477 **($C_8H_{18}O_2PS_2Na$)** Dithiophosphate	50 50 50	29%

Table 5.3 Frother dosages used in all flotation tests.

Flotation Stage	Frother	Dosage (mg)
1st flotation stage	Pine Oil	100
	MIBC	60
	Aero 65 (prolypropylene glycol)	50
2nd flotation stage	Pine Oil	50
	MIBC	60
	Aero 65 (polypropylene glycol)	50

5.3.1 Preparation of flotation products

Filtration was carried on the various flotation products to dewater them. Both the filtered flotation concentrates and tailings were dried for 24 hours in a laboratory oven. A Fisher Furnace, Model 490 (Coal Analyzer Furnace), was used to ash dried filter papers. Filter paper ash was added to the respective, dried flotation product prior to analytical sample preparation. Figure 5.2 shows the protocol of flotation experiments and flotation products preparation and analysis.

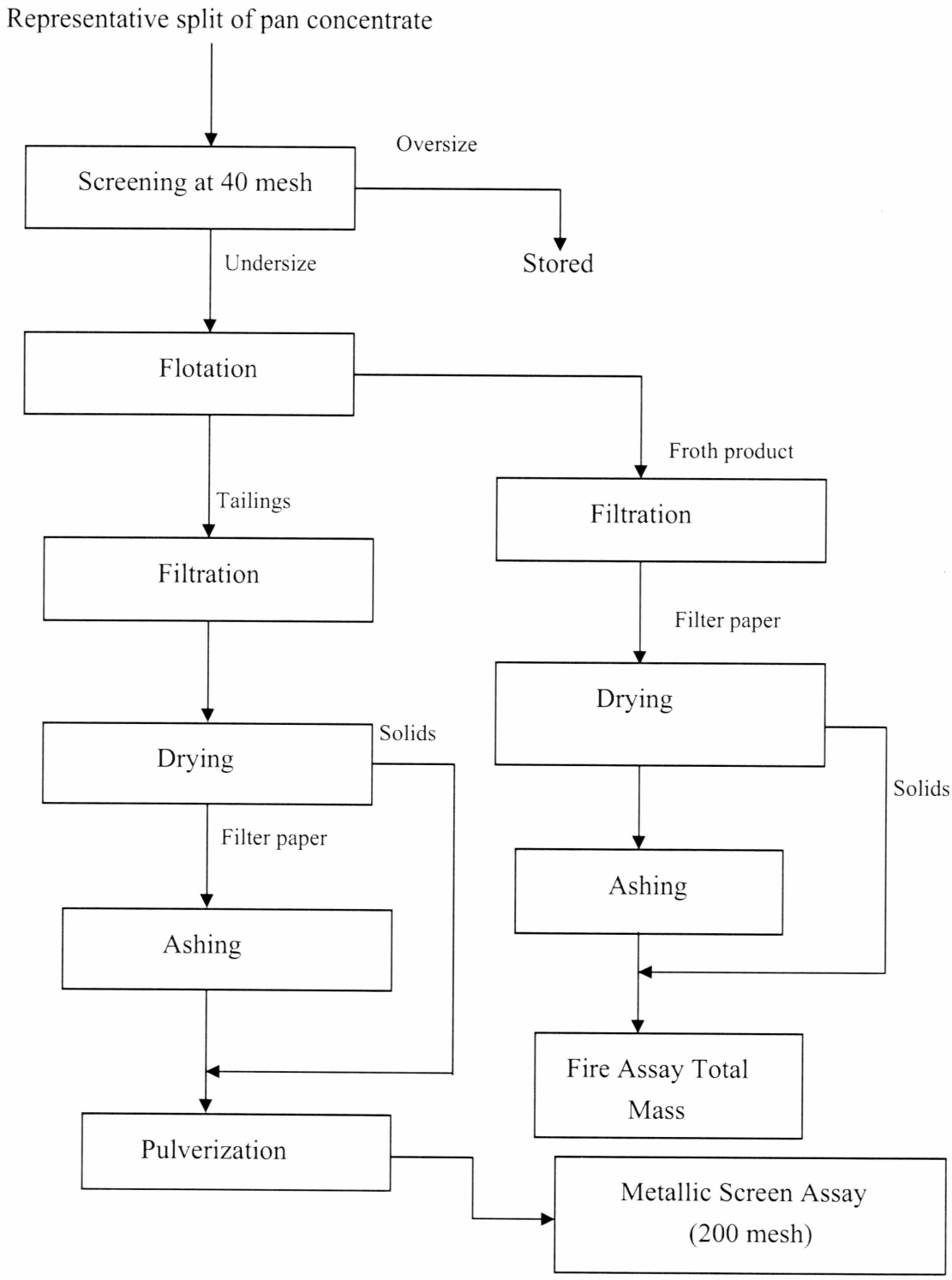


Figure 5.2 Flowsheet of laboratory flotation product preparation and analysis.

5.3.2 Analytical sample preparation and analysis

A Rock Labs ring and puck mill was employed to pulverize products ahead of analyses. A metallic screen assay procedure was used, due to the particulate nature of the placer gold and PGM mineralogy present. Early metallic screen assays utilized a 100 mesh (149 microns) screen, but this screen size was subsequently reduced to 200 mesh (74 microns) based on early results. Flotation concentrates and some other low mass (less than 30 grams) products leant themselves to total mass fire assays, followed by ICP analysis for gold and platinum values. For larger samples (several hundreds grams), the metallic screen procedure was used. Plus screen material received a total mass fire assay, followed by ICP analysis for gold and platinum values. Minus screen material received duplicate 30 grams fire assays followed by ICP analysis for gold and platinum values. All gold and platinum analyses were performed under contract by ALS CHEMEX, through their Vancouver, B.C., Canada laboratory.

The higher grade sample were treated by ALS CHEMEX procedures Au CON01A and Pt CON01A at a cost of $50 per element (gold and platinum analyses cost $100 per sample). Low grade samples were treated by ALS CHEMEX procedure PGMICP27, at a cost of $15 per analysis (duplicate analysis cost $30).

5.4 Kinetics study

After a critical analysis of the assay results of the flotation studies, a kinetics study was carried out using the reagents, which gave the best recovery of platinum from the marine

sediments. The aim of kinetics study was to investigate the rate of platinum and gold flotation. The study was carried out as described in the following procedure:

1) Solid sample and water were added to the 2 liter flotation cell.
2) Agitation was initiated at 1300 rpm.
3) An attrition scrubbing period of 5 minutes was allowed.
4) Aero 350 xanthate and Aero 3477 dithiophosphate collectors were introduced into the cell at a dosages of 100 mg/kg and 50 mg/kg, respectively.
5) A conditioning period of ten minutes was allowed.
6) Stage 1 frothers (Table 5.3) were introduced to the cell.
7) A conditioning period of five minutes was allowed.
8) A baffle was inserted onto the flotation cell and pressurized air flow was introduced at a rate of 6-6.5 l/min.
9) 1st frother product was collected from 0 to 2 minutes.
10) 2nd froth product was collected from 2 to 4 minutes.
11) 3rd froth product was collected from 4 to 6 minutes.
12) 4th froth product was collected from 6 to 8 minutes.
13) 5th froth product was collected from 8 to 10 minutes.
14) Stage 2 frothers (Table 5.3) were introduced into the cell.
15) 6th froth product was collected from 10 to 13 minutes.
16) 7th froth product was collected from 13 to 18 minutes.
17) 8th froth product was collected from 18 to 23 minutes.

18) Air and agitation were shut off and concentrate and tailings samples were prepared for analysis.

5.5 X-ray fluorescence

In order to estimate the elemental composition of the major oxides associated with the marine sediments, X-ray fluorescence (XRF) analysis of the pan concentrate was conducted.

Press pellets were made from the following materials:

- A representative split of pan concentrate.
- +40 mesh pan concentrate.
- A low intensity, non magnetic fraction of pan concentrate.
- –40 mesh pan concentrate.

A gram of each of the above products was weighed and ground in an agate mortar for 3 minute. The finely ground sample was mixed with four drops of polyvinyl alcohol dissolved in water (20/100 ml H_2O) and binder and ground again for 2 minute. The mixed sample and binder were transferred to a Craver laboratory hydraulic press and compressed under a 2,000 pound force for 2-3 minutes.

An Axios Wavelength Dispersive Spectrometer (WDS) manufactured by the PaNalytical, an X-ray analytical microprocessor instrument equipped with Rhodium (Rh) target and

X-ray tube, with anode voltage of 60 kV above the ground potential, was used for XRF analysis. XRF emitted from the near surface part of the sample is detected by a wavelength dispersive detector with a beryllium window of 75 μm. The whole system is controlled from an external computer, running SuperQ software package. A standard routine and application which contains the elemental composition of the sample was created. Calibration of the Axios WDS was performed to determine the sensitivity of spectrometer and line overlap factors by using the standards.

Analytical recipes and set of parameters were defined for the standards. The spectrometer was then calibrated by measuring the set of standards with a tube voltage of 60 kV and tube current of 66 mA. The calibration sequence consists of the peak search, peak match and calibration using the standards. XRF intensity measurements of the pressed pellets were then conducted using the calibrated Axios WDS. The lower limits of detection and the characteristic lines were also determined. Results of the XRF analysis of the pan concentrate materials are presented in Chapter 6.

5.6 Microprobe analysis of PGM and gold mineralogy

A flotation test identical in protocol to the flotation test 'I' was carried out, in order to produce a flotation froth product for microprobe analysis of its PGM and gold mineralogy. Only initial, preliminary results were obtained; future work by others will build upon this cursory, initial work. Using the University of Alaska Fairbanks

microprobe operated by Dr. Ken Severin, X-ray spectra were acquired with a Edax Phoenix energy dispersive system on a Cameca –SX50. Semi-quantitative (standardless) results were obtained using Edax Quantzaf 3.34 software.

Chapter 6 Results and Discussion

6.1 Results of low intensity, magnetic separation

The results of the low intensity, magnetic separation test are presented in Tables 6.1-6.4. The pan concentrate is composed of a significant amount of low intensity magnetic mineral, likely magnetite, approximately 49% by weight.

Table 6.1 First stage, low intensity, magnetic separation test conditions and results.

Weight of feed	1110.8 g
Feed control	45-50%
Rotor speed	50 rpm
Gap (clearance)	12 mm
Required time for separation	19 min
Non-magnetic product	622.4 g
Magnetic product	487.6 g
Actual feed rate	58 g/min

Table 6.2 Second stage, low intensity, magnetic separation test conditions and results.

Weight of feed	622.4 g
Feed control	45-51 %
Motor speed	50 rpm
Gap (clearance)	6 mm
Required time for separation	11 min
Non-magnetic product	575.8 g
Magnetic product	46.4 g
Actual feed rate	57 g/min

Table 6.3 Third stage, low intensity, magnetic separation test conditions and results.

Weight of feed	575.8 g
Feed control	45-50 %
Rotor speed	50 rpm
Gap (clearance)	3 mm
Required time for separation	8 min
Non-magnetic product	567.4 g
Magnetic product	8.3 g
Actual feed rate	72 g/min

More importantly, gold and platinum analytical results of the low intensity, magnetic separation products reveal that the magnetic material from the various stages of separation does not contains gold, and contains very little of platinum. It is likely that the

platinum reporting to the magnetic fractions exists as unliberated PGM mineralogy associated with magnetite grains, which could not be concentrated efficiently in a placer processing plant by either gravity concentration or froth flotation. 100% of the gold values and 97% of the platinum values reported to the non magnetic fractions, during low intensity, magnetic separation. This result implies that low intensity magnetic separation may be used efficiently in a flowsheet to process marine sediments from this area, in order to reduce material bulk ahead of the gravity concentration and froth flotation processes.

Table 6.4 Analyses of products from the low intensity, magnetic separation test.

Product	wt (g)	Assay (mg/kg)		Metal Content (mg)		Distribution %	
		Au	Pt	Au	Pt	Au	Pt
Stage 1 Magnetics	487.60	0	0.06	0	0.03	0	2.5
Stage 2 Magnetics	46.40	0	0.13	0	0.01	0	0.7
Stage 3 Magnetics	8.30	0	0	0	0	0	0
Stage 3 Non magnetics	567.40	0.9	2.04	0.51	1.16	100.0	96.7
Total	1109.70	0.46	1.28	0.51	1.20	100.0	100.0

6.2 Results of particle size analysis and microprobe analysis

Results of the particle size analysis are presented in Tables 6.5-6.6. The gold and platinum distributions confirm what was observed microscopically, at various stages throughout this study:

1) The placer gold is coarser than the placer platinum.
2) The placer gold exists, for the most part, within the 50×150 mesh size fraction.
3) The placer platinum exists, for most part, within the 100×200 mesh size fraction.

Similar to the magnetic separation test results, platinum values analyzed in the coarser size fractions, are very likely the results of unliberated PGM values. Not shown by the particle size analysis, is the significant difference in particle shape observed microscopically, between placer gold and placer PGM particles. PGM particles have a much higher Corey's shape factor[46] than the placer gold; that is, the PGM grains are more spherical than the gold grains. Though no statistically valid measurements were made, microscopic observations estimate the Corey's shape factors for the PGM and gold grains at 0.7-0.9 and 0.2-0.3, respectively. The flakey nature of the placer gold grains is consistent with their proposed provenance; originating from further east and eroded and transported by glacial action[2]. Preliminary, qualitative, microprobe analysis of PGM grains from flotation concentrate showed grains of nearly pure iridium, isoferroplatinum, and Pt-Rh-Ir-Fe-S-As mineralogy.

Table 6.5 Size distribution data for representative pan concentrate split.

Sieve size range (μ m)	Sieve fraction		Cumulative % Undersize	Cumulative % Oversize
	wt (g)	wt (%)		
850×600	33.50	3.13	96.87	3.13
600×425	131.72	12.31	84.56	15.44
425×300	344.39	32.20	52.36	47.64
300×212	397.36	3 7.15	15.21	84.79
212×150	141.85	13.26	1.95	98.05
150×106	18.85	1.76	0.19	99.81
106×75	1.68	0.16	0.08	99.97
75×53	0.13	0.01	0.02	99.98
53×38	0.11	0.01	0.01	99.99
-38	0.11	0.01		
Total	1069.70	100.00		

Table 6.6 Analyses of size fractions for gold and platinum.

ASTM Mesh	Size Fraction (μ m)	wt (g)	Assay (mg/kg)		Metal Content (mg)		Distribution (%)	
			Au	Pt	Au	Pt	Au	Pt
20×30	850×600	33.50	0	0	0	0	0	0
30×40	600×425	131.72	0	0	0	0	0	0
40×50	425×300	344.39	0	0.09	0	0.03	0	2.8
50×70	300×212	397.36	0.18	0.25	0.07	0.10	10.6	9.2
70×100	212×150	141.85	1.27	0.14	0.18	0.02	27.3	1.9
100×140	150×106	18.85	18.04	18.56	0.34	0.35	51.5	32.4
140×200	106×75	1.68	29.76	303.57	0.05	0.51	7.6	47.2
-200	-75	0.35	57.14	200.00	0.02	0.07	3.0	6.5
Total		1069.70	0.62	1.01	0.66	1.08	100.0	100.0

6.3 Froth flotation results

The purpose of the flotation tests was to screen a variety of collectors known to be effective for gold flotation, for their effectiveness on PGM minerals found in the

Goodnews Bay marine sediments. Only the -40 mesh size fraction of the pan concentrate splits was processed for the flotation tests, which were conducted at neutral pH.

Pulp densities of the flotation experiment were in the range of 27-30% solids by weight. All reagents added to flotation were in liquid form and the flotation procedure was carried out at room temperature. Results of the flotation test are presented in Tables 6.7–6.13. The data show that, as expected, the investigated collectors all provided high gold recoveries, 82.7% to 99.8%. However, their effectiveness as collectors for the platinum mineralogy varied drastically, yielding platinum recoveries over a broad range, 4.1% to 80.4%.

Individually, Aero 350 (potassium amyl xanthate) produced the highest platinum recovery, S-8474 (diakyl dithiocarbamate) showed some promise, but Aero 3477 (sodium isobutyl dithiophosphate), Aero 3320 (xanthate allyl ester), and Aero 5100 (allyl alkyl thinonocarbamate) all yielded very poor platinum recoveries. A combination of Aero 350 and Aero 3477 (Table 6.12) produced a 75% platinum recovery, likely due to the presence of the xanthate, though the dithiophosphate demonstrated excellent froth stabilization characteristics. A combination of xanthate, dithiophosphate and dithiocarbamate (Table 6.13) gave poorer recovery of platinum and gold as compared to a combination of xanthate and dithiophosphate (Table 6.12). Future studies should focus on Aero 350 dosage optimization and investigating other xanthate collectors.

Table 6.7 Froth flotation test 'C' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	9.0	90.92	108.13	0.818	0.973	82.7	80.4
Tailings							
Met Screen (+200 mesh)	28.1	1.82	2.53	0.051	0.071	5.2	5.9
Met Screen (-200 mesh)	752.6	0.16	0.22	0.120	0.166	12.1	13.7
Calc. feed	789.6	1.25	1.53	0.989	1.21	100.0	100.0

Reagent dosage:

Collector: Aero 350 Xanthate = 100 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

Table 6.8 Froth flotation test 'D' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	9.9	45.48	28.51	0.450	0.282	92.2	24.6
Tailings							
Met Screen (+200 mesh)	25.1	1.52	24.15	0.038	0.606	7.8	52.8
Met Screen (-200 mesh)	810.1	0.0	0.32	0.0	0.259	0.0	22.6
Calc. feed	845.1	0.58	1.36	0.488	1.147	100.0	100.0

Reagent dosage:

Collector: S-8474 Dithiocarbamate = 100 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

Table 6.9 Froth flotation test 'E' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	11.8	43.11	3.57	0.509	0.042	92.5	4.1
Tailings							
Met Screen (+200 mesh)	24.8	0.73	27.58	0.018	0.684	3.3	66.7
Met Screen (-200 mesh)	765.8	0.03	0.39	0.023	0.299	4.2	29.2
Calc. feed	802.4	0.69	1.28	0.550	1.025	100.0	100.0

Reagent dosage:

Collector: Aero 3477 Dithiophosphate = 100 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

Table 6.10 Froth flotation test 'F' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	11.9	44.86	3.54	0.534	0.042	99.8	4.3
Tailings							
Met Screen (+200 mesh)	23.1	0.07	19.62	0.001	0.453	0.20	46.5
Met Screen (-200 mesh)	761.3	0.00	0.63	0.000	0.480	0.0	49.2
Calc. feed	796.3	0.67	1.22	0.535	0.975	100.0	100.0

Reagent dosage:

Collector: Aero 3320 Xanthate Ester = 100 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

Table 6.11 Froth flotation test 'G' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	12.2	30.08	10.32	0.367	0.126	63.9	10.9
Tailings							
Met Screen (+200 mesh)	20.7	4.47	33.21	0.093	0.687	16.2	59.4
Met Screen (-200 mesh)	715.6	0.16	0.48	0.114	0.343	19.9	27.7
Calc. feed	748.5	0.77	1.54	0.574	1.156	100.0	100.0

Reagent dosage:

Collector: Aero 5100 Thionocarbamate = 100 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

Table 6.12 Froth flotation test 'I' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
1st stage conc.	6.2	75.85	150.51	0.470	0.933	87.5	74.6
2nd stage conc.	11.4	5.76	0.74	0.066	0.008	12.3	0.6
Tailings							
Met Screen (+200 mesh)	20.4	0.07	4.52	0.001	0.092	0.2	7.4
Met Screen (-200 mesh)	833.6	0.0	0.26	0.0	0.217	0.0	17.4
Calc. feed	871.6	0.62	1.43	0.537	1.25	100.0	100.0

Reagent dosage:

Collectors: Aero 350 Xanthate = 50 mg/kg;

Aero 3477 Dithiophosphate = 50 mg/kg.

1st stage frothers: Pine Oil = 100 mg; MIBC = 60 mg; Aero 65 = 50 mg.

2nd stage frothers: Pine Oil = 50 mg; MIBC = 60 mg; Aero 65 = 50 mg

Table 6.13 Froth flotation test 'L' results.

Sample	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
Flotation Conc.	12.6	63.57	44.10	0.801	0.556	93.4	56.3
Tailings							
Met Screen (+200 mesh)	24.9	1.61	11.97	0.040	0.298	4.7	30.2
Met Screen (-200 mesh)	789.6	0.02	0.17	0.016	0.134	1.9	13.5
Calc. feed	827.1	1.04	1.19	0.857	0.998	100.0	100.0

Reagent dosage:

Collectors: Aero 350 Xanthate = 50 mg/kg.
S- 8474 Reagent Dithiocarbamate = 50 mg/kg.
Aero 3477 Dithiophosphate = 50 mg/kg.

Frothers: Pine Oil = 150 mg; MIBC = 120 mg; Aero 65 = 100 mg.

6.3.1 Froth characteristics

Observing the froth characteristics and the stability of the froth during the flotation experiments involving dithiophosphate, one would think that dithiophosphate would give a good recovery of the platinum and gold. Dithiophosphate alone gave excellent froth

characteristics throughout the entire flotation experiment. The depth of the froth and density of the froth increased with the addition of pine oil and MIBC. Froth bubbles did not break down during froth collection, as was the case with other collectors. The addition of Aero 65 did not change the frothing characteristics.

Potassium amyl xanthate, on the other hand, presented different frothing characteristics all together. There was no significant change in the froth characteristics after addition of the xanthate. Observation of the froth, with only xanthate present as a reagent, revealed that the formed bubbles broke very rapidly, yielding an unstable froth. This condition changed upon addition of the frothers, pine oil and MIBC; the frothing characteristics became very stable, with a sharp increase in the froth thickness. The addition of Aero 65 (polypropylene glycol) increased the froth stability. Froth stability in the flotation of these marine sediments, whose flotable metals values are an extremely low percentage of the feed material, will be a very important factor for overall flotation performance.

6.4 Results of flotation kinetics study of platinum and gold

Based on the superior collector performance of the Aero 350 (potassium amyl xanthate) and the frothing characteristics of the Aero 3477 (isobutyl dithiophosphate), it was decided to conduct an additional flotation test with these collectors, and at the same time investigate the kinetics of the gold and platinum flotation. Table 6.14 and Figure 6.1 show the results. Equilibrium gold recovery compares well with previous xanthate and dithiophosphate flotation tests (Tables 6.7, 6.9 and 6.12). However the low equilibrium

platinum recovery of 46.7% was significantly below the previous flotation tests where xanthate was employed, (Tables 6.7, 6.12 and 6.13). Perhaps this lower platinum flotation recovery is a result of particle lock-up in the flotation cell, analytical error, sample preparation error or another cause. Nevertheless, the kinetics study shows that the response of the floatable gold and platinum in this sample was rapid, with equilibrium platinum recovery being achieved within 10 minutes and equilibrium gold recovery being achieved in 20 minutes.

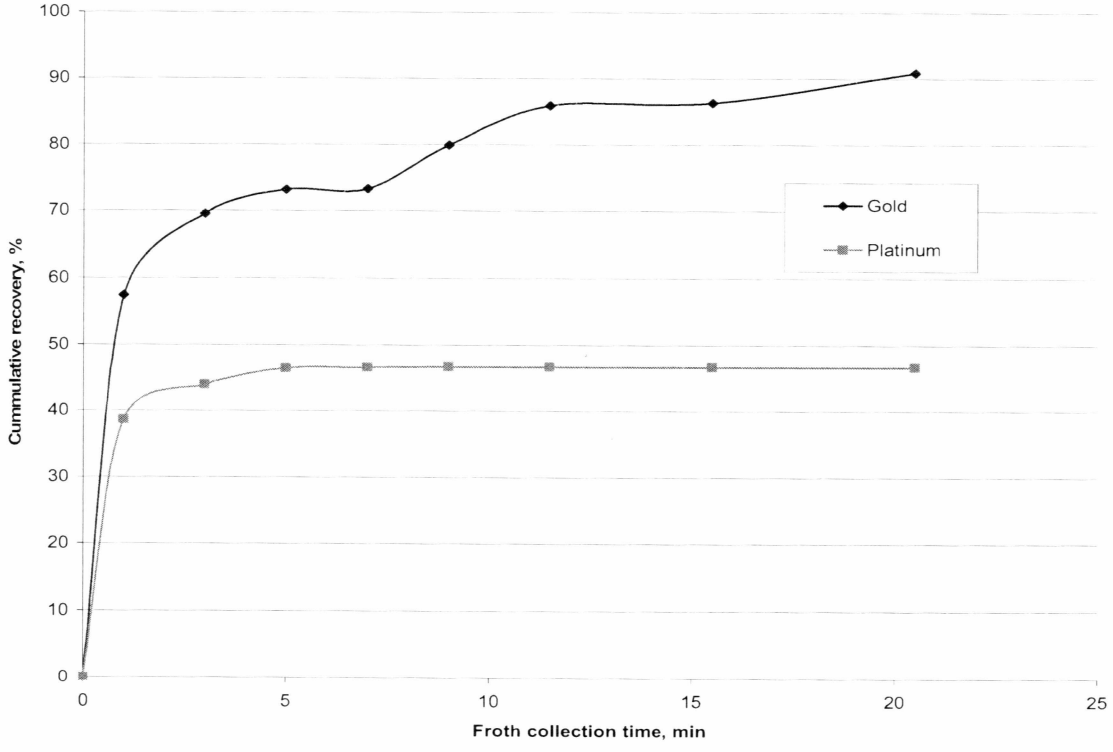


Figure 6.1 Cumulative recoveries of gold and platinum with time.

Table 6.14 Results of kinetics flotation test.

Froth Collection Time (min)	wt (g)	Assays (mg/kg)		Metal Content (mg)		Distribution (%)	
		Au	Pt	Au	Pt	Au	Pt
0-2	2.4	91.4	222.72	0.218	0.535	57.4	38.7
2-4	1.8	25.76	40.37	0.046	0.073	12.1	5.3
4-6	0.5	27.77	67.39	0.014	0.034	3.7	2.5
6-8	0.3	1.16	6.16	0.0003	0.002	0.1	0.1
8-10	0.6	39.54	3.68	0.024	0.002	6.3	0.1
10-13	1.8	13.53	0.07	0.024	0.0001	6.3	0.0
13-18	1.2	0.64	0.08	0.001	0.0001	0.3	0.0
18-23	1.6	11.43	0.12	0.018	0.0002	4.7	0
Tailings							
Met Screen (+200 mesh)	26.8	0.08	12.56	0.002	0.337	0.5	24.3
Met Screen (-200 mesh)	835.2	0.04	0.48	0.033	0.401	8.6	29
Calc. feed	872.2	0.44	1.59	0.38	1.384	100	100

Reagent dosage:

Collectors: Aero 350 Xanthate = 100 mg/kg.
Aero 3477 Dithiophosphate = 50 mg/kg.

Frothers: Pine Oil = 100 mg; MIBC = 60 mg; Aero 65 = 50 mg.

6.5 Results of X-ray fluorescence analysis

The elemental compositions of the major oxides present in the marine sediment were analyzed using the K-α lines. Elements whose emission energy is greater the energy of the spectrometer were analyzed with the L-α lines. Figure 6.2, Figure 6.3 and Figure 6.4 show the calibration curves of aluminum oxide, zirconium and iron oxide, respectively. Table 6.15 shows the quantitative XRF analysis of major oxides and other elements in the pan concentrate sample.

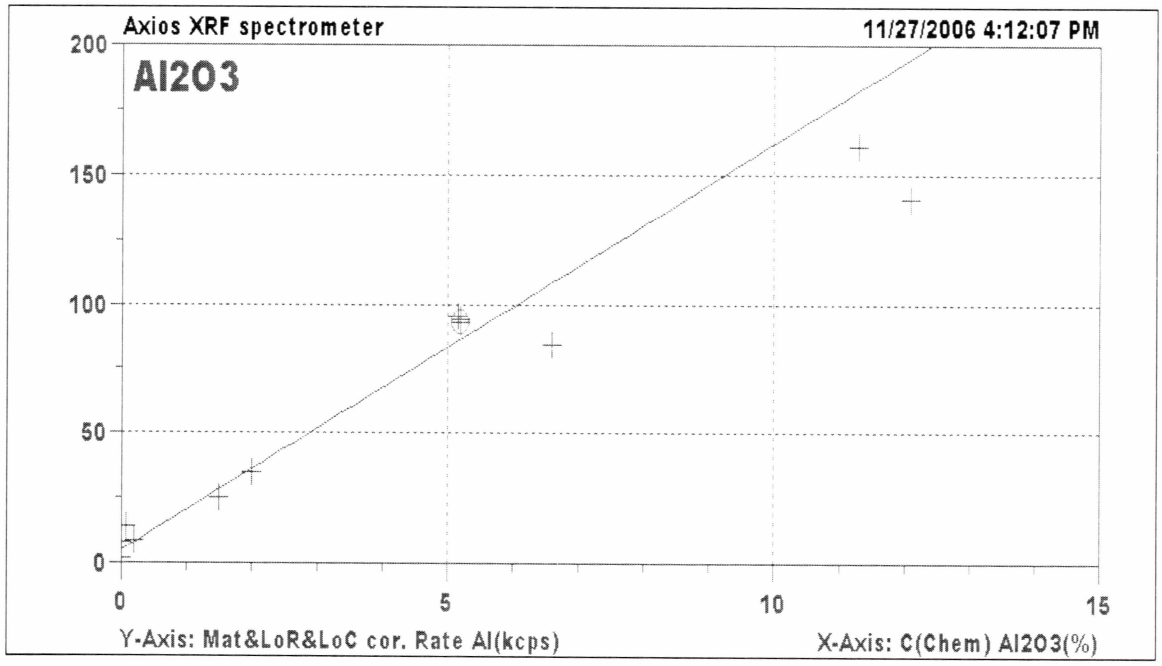


Figure 6.2 Calibration curve for aluminum oxide.

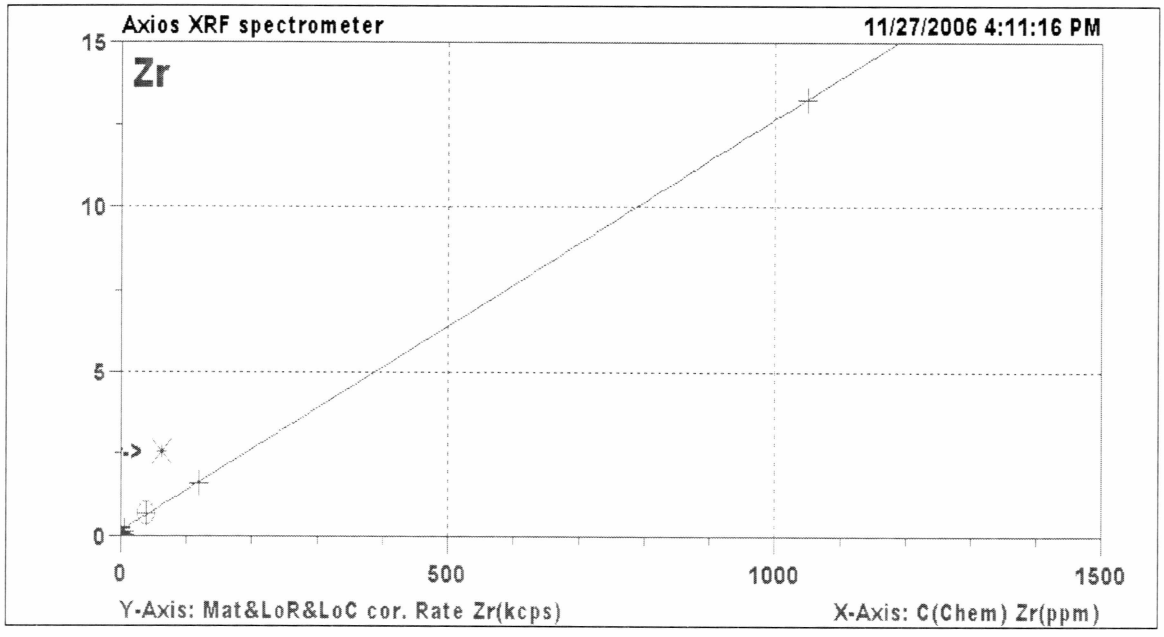


Figure 6.3 Calibration curve for zirconium.

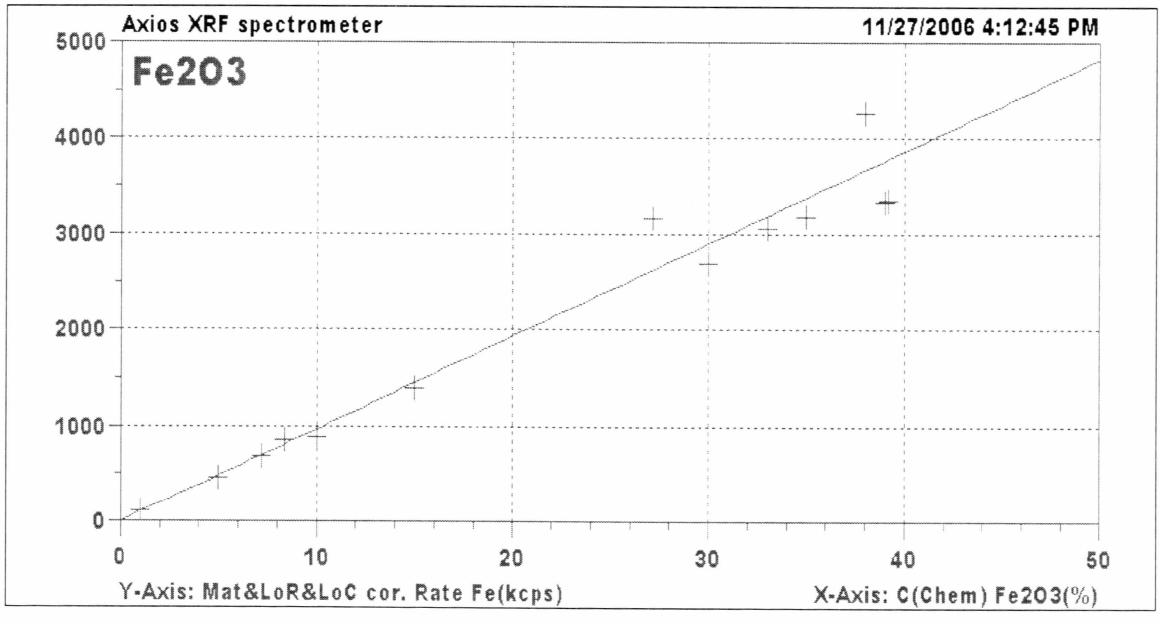


Figure 6.4 Calibration curve for iron oxide.

Table 6.15 Results of XRF analysis of pan concentrate materials

Pellet No.	Fe_2O_3 %	SiO_2 %	Al_2O_3 %	MgO %	Cr %	CaO %	MnO %	Zn ppm	Co ppm	Sr ppm	Zr ppm
PFP-1	36.5	21.0	6.2	6.2	5.9	3.3	0.29	389	180	150	42
PFP-2	48.9	23.3	8.2	7.9	8.9	3.7	0.34	547	244	78	116
PFP-3	30.3	39.8	10.2	8.5	4.3	8.9	0.23	258	145	164	51
PFP-4	43.4	16.4	5.9	6.5	6.5	2.6	0.29	488	198	97	32

PFP-1: Representative split of pan concentrate.

PFP-2: +40 mesh pan concentrate.

PFP-3: Low intensity, non magnetic fraction of pan concentrate

PFP-4: -40 mesh pan concentrate

It should be noted that non-oxide elements like gold, platinum and others are excluded from Table 6.15.

Chapter 7 Metallurgical Flowsheet Design and Cost Estimation

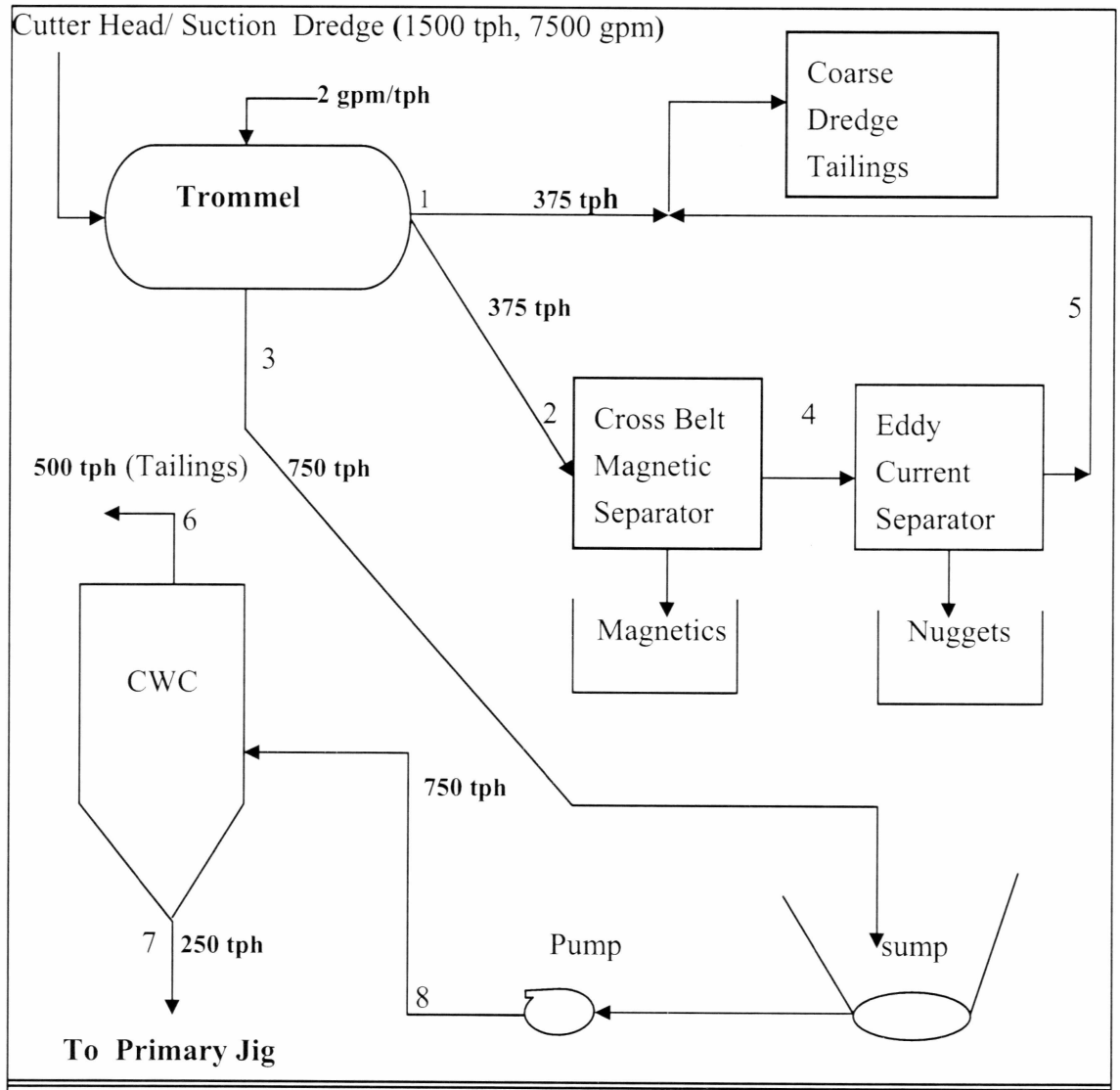
By utilizing the metallurgical results of this study, previously determined gravity concentrator performance versus precious metals size and shape[6,7], and mineral processing engineering data, a precious metals recovery plant was designed for a marine placer located west of Red Mountain, Alaska. The processing plant was designed to treat 1500 tons per hour of marine placer material from a dredge operation.

Placer marine sediments from the dredge operation are fed to a trommel, where the material is washed and screened. The +2 inch, oversize material from the trommel would be discharged at rear side of the dredge. The $2\times3/8$ inch material would be sent to a crossbelt magnetic separator, then to an eddy current separator as an option to recover gold and platinum nuggets[47]. Though shown as a flowsheet option, the cross belt magnetic separator and the eddy current separator are not included in the cost estimates. Conceptually, scrubbing, screening and three stages of gravity concentration (preconcentration and two stages of jigging) would occur on the dredge platform, allowing the majority of the tailings to be discharged back into the marine environment as part of dredging operation. Secondary jig concentrate is pumped onshore for upgrading via additional stages of gravity concentration and froth flotation. Fine (-50 mesh) jig tailings are also pumped onshore for processing by froth flotation. Froth flotation tailings are impounded in a tailings facility onshore.

Mass and water balances around unit processes circuits were determined using the metallurgical size distribution data shown in Table 7.1[48] and accepted mineral processing engineering techniques. By employing accepted metallurgical engineering design techniques[49,50] and experience, the mass and water balance were used to select various processing units for the plant: the scrubbing, screening and preconcentration section, the gravity concentration section and the froth flotation section. The proposed flowsheets for each section are shown in Figure 7.1, Figure 7.2 and Figure 7.3, respectively.

Table 7.1 Size distribution used for mass balances[48].

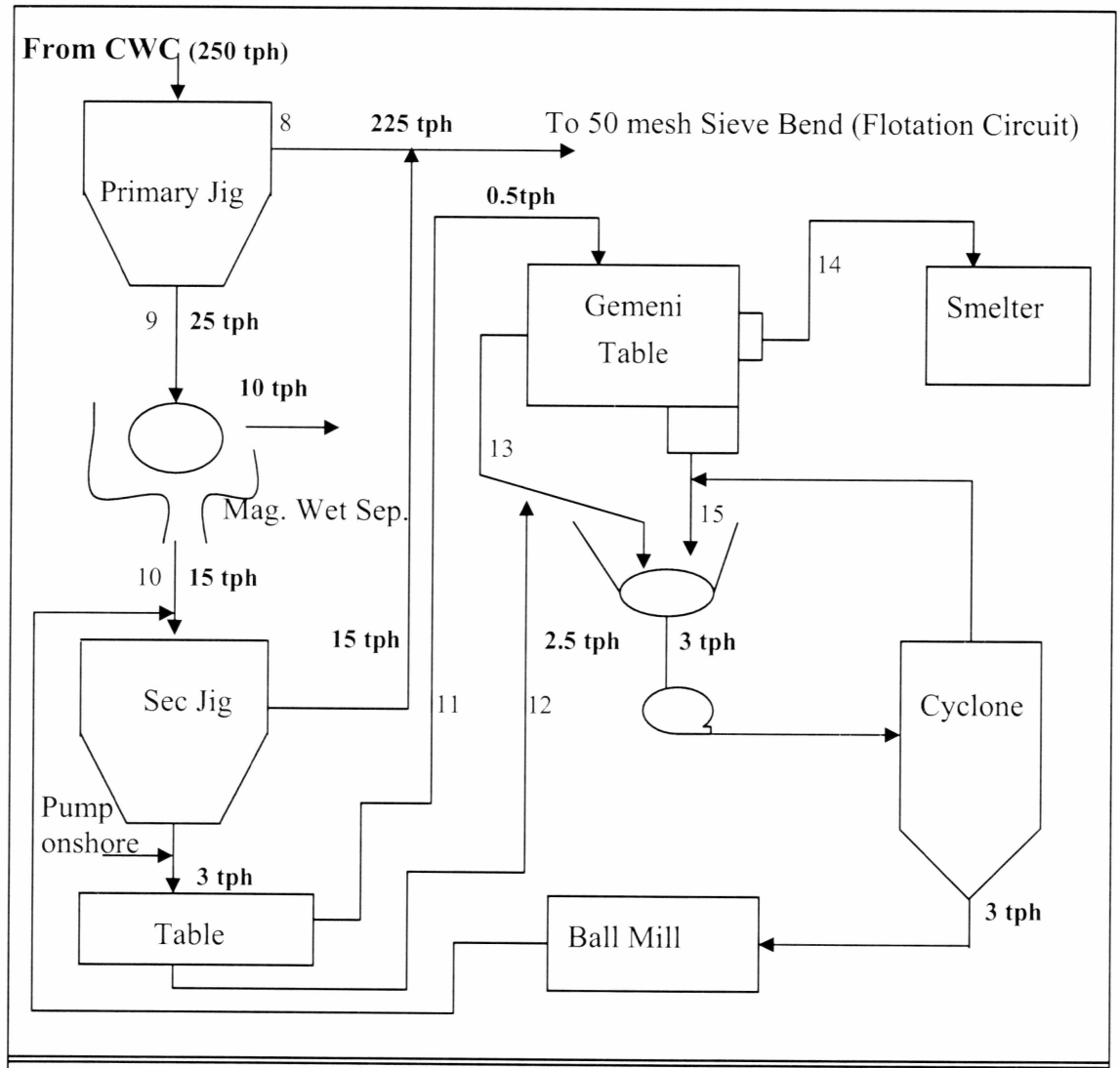
Size Fraction (inches)	**Weight %**
+2	25
$2\times3/8$	25
$-3/8$	50
-50 mesh	30



Stream Identification

1. + 2 inch material
2. 2×3/8 inch material
3. - 3/8 inch material
4. Non magnetic material
5. 2×3/8 inch Tailings
6. CWC overflow (Fine Dredge Tailings)
7. CWC underflow
8. Pumped -3/8 inch feed from sump

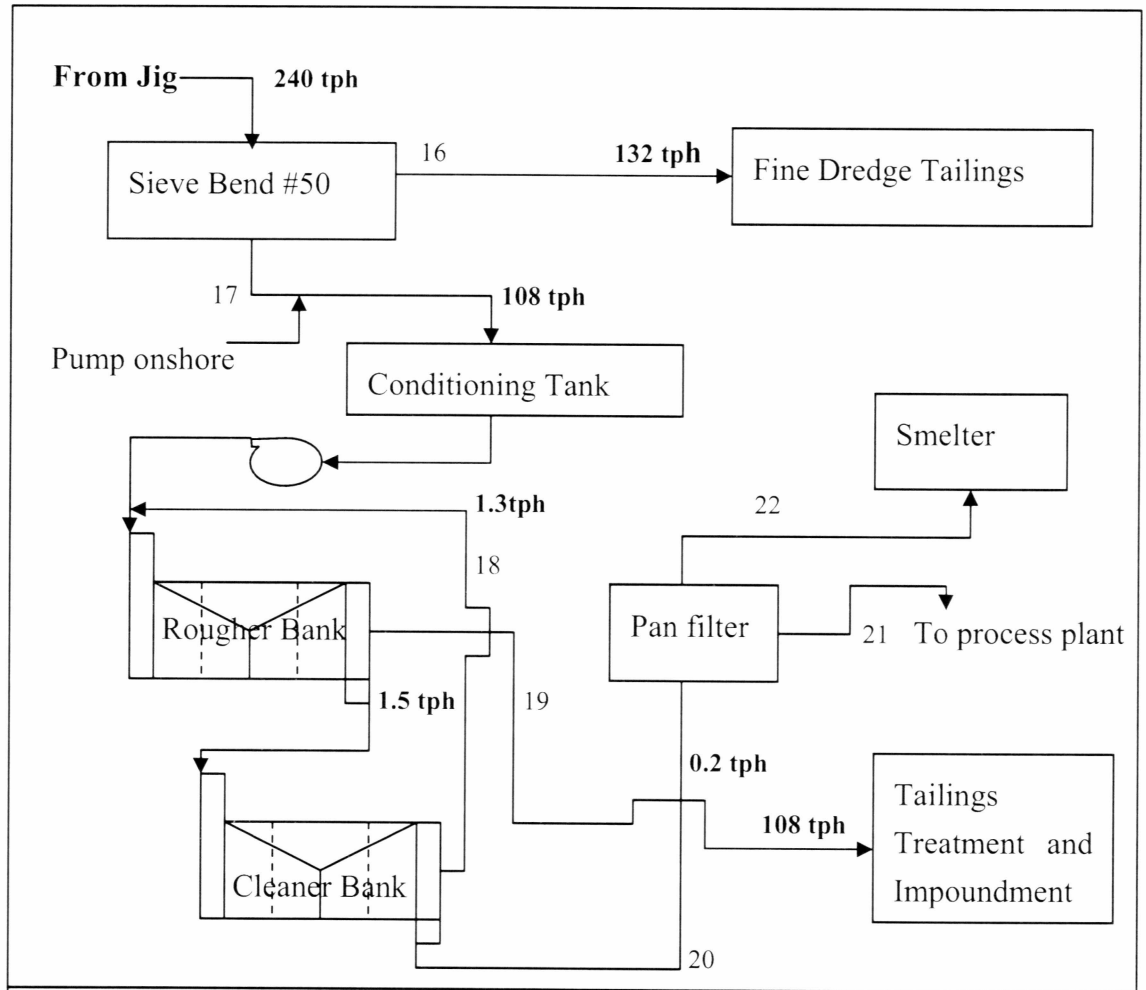
Figure 7.1 Scrubbing, screening and pre-concentration section flowsheet.



Stream Identification

8. Jig Tails
9. Jig Concentrate
10. Non Magnetics Material
11. Table Concentrate
12. Table Tailings
13. Gemeni Table Tailings
14. Gemeni Table Concentrate
15. Gemeni Table Middlings

Figure 7.2 Gravity concentration section flowsheet.



Stream Identification

16. + 50 mesh material
17. - 50 mesh material
18. Cleaner Tails
19. Rougher Tails
20. Flotation Concentrate
21. Filtrate
22. Filter Cake

Figure 7.3 Froth flotation section flowsheet.

Western Mine Engineering's Mine and Milling Equipment Cost Guide (WME)[51], the Coal Cleaning Cost Model (CCCM)[52], accepted mineral processing techniques[50,51], and quotes for capital costs from some manufacturers of mining equipment were used to estimate the capital costs for all the processing equipment. Table 7.2, Table 7.3 and Table 7.4 provides a summary of the estimates for equipment capital costs, specifications for the equipment and electrical power requirements for the process equipment.

Estimated operating costs were based on the approached outlined in the Coal Cleaning Cost Model[52] and adjusted to 2006 US dollars, using appropriate cost indexes supplied by Western Mine Engineering's Mine and Milling Equipment Cost Guide[51]. Operating costs were first estimated for 100% processing plant availability, and then adjusted for several availabilities ranging from 50% to 90%. The coast of Southwestern Alaska is a harsh environment and actual dredging availability is uncertain, but expected to be impacted by winter conditions, storm events and other weather factors.

Both capital cost estimates and operating cost estimates consider a 'location factor' for an operation near Goodnews Bay, Alaska. Based on advice from Great Northern Engineering (GNE) Palmer, Alaska, an R.S. Means [53] estimation approach was used[54]. The R.S. Means approach is based on a base case, US average location, being assigned an index value of 100. Hence, an R.S. Means index of 150 would imply 50% higher construction costs than the US average location.

An R.S. Means index of 50, would imply 50% lower costs than the U.S average location. Great Northern Engineering notes that Anchorage, Alaska has R.S. Means indexes of 130 for labor and 125 for materials. The indexes that GNE uses for Dillingham, Alaska (approximately 125 air miles east of Platinum, Alaska) are 205 for labor and 200 for materials. GNE estimates that the indexes for the Goodnews Bay, Alaska location may be about 5% higher than those for Dillingham, Alaska, in order to cover the extra shipping costs. Based on this information, it was decided that to use an R.S. Means index of 215 for the Goodnews Bay, Alaska area for all costs (capital equipment, material and supplies, labor etc.), or a location factor of (215/100) = 2.15.

Table 7.5 to Table 7.12 show estimates of the operating costs for the marine placer plant with 100% availability . Table 7.13 to Table 7.17 show estimates of the operating costs for the marine placer plant with availability factors; 50%, 60%, 70%, 80%, and 90%. Material and supplies were estimated at 10% of the total equipment capital cost.

Table 7.2 Capital equipment cost estimate for the scrubbing, screening and pre-concentration section of the flowsheet.

Equip#	Description	Source	Capacity	Hp	Price	Qty.	2006 $
E100	Trommel	CEH	750 tph		$412,000	2	$ 824,000
E101	Motor	WME		100	$ 7,950	2	$ 15,900
E200	Main Pump	WME	5000 gpm		$ 44,100	2	$ 88,200
E201	Motor	WME		250	$ 20,050	2	$ 40,100
E300	26" CWC	WME	750 tph		$ 26,000	8	$ 208,000
E400	Primary Jig 6 Cell Radial	WME	250 tph	7.5	$219,000	1	$ 219,000
E401	Motor		Included in E400				
E402	Pump	WME	807 gpm		$ 12,900	1	$ 12,900
E403	Motor	WME		25	$ 2,229	1	$ 2,229
E404	Pump	WME	2356 gpm		$ 25,900	1	$ 25,900
E405	Motor	WME		50	$ 4,784	1	$ 4,784
E500	Sieve Bend	WME	4113 gpm		$ 31,600	3	$ 94,800
E501	Motor	WME		3	Included in E500		
Subtotal							**$ 1,535,813**

Table 7.3 Capital equipment cost estimate for the gravity concentration section of the flowsheet.

Equip#	Description	Source	Capacity	Hp	Price	Qty	2006 $
E600	Wet Magnetic Separator	WME	81 gpm	1.5	$ 10,000	1	$ 10,000
E601	Motor	WME		Included in E600			
E700	Sec. Jig 1 cell Radial	WME	15 tph	3	$ 14,800	2	$ 29,600
E701	Motor	WME		Included in E700			
E702	Pump	WME	180 gpm		$ 8,700	1	$ 8,700
E703	Motor	WME		5	$ 940	1	$ 940
E800	Conc. Table	WME	3 tph	2	$ 32,600	2	$ 65,200
E801	Motor	WME	Included in E800				
E900	Gemeni Table	KGS	1000 lb/hr	3	$ 30,000	2	$ 60,000
E901	Motor	WME	807 gpm	Included in E900			
E1000	Cyclone	WME	51 gpm		$ 1,710	2	$ 3,420
E1001	Pump	WME			$ 8,700	1	$ 8,700
E1002	Motor	WME		2.5	$ 470	1	$ 470
E1100	Ball Mill	WME	3 tph		$256,000	1	$ 256,000
E1101	Motor	WME		40	$ 2,786	1	$ 2,786
Subtotal							**$ 445,816**

Table 7.4 Capital equipment cost estimate for the froth flotation section of the flowsheet.

Equip#	Description	Source	Capacity	Hp	Price	Qty	2006 $
E1200	Flotation (Rougher)	WME	500 cu.ft		$ 53,600	16	$857,600
E1201	Motor	WME		40	$2,786	16	$ 44,576
E1202	Pump	WME	1850 gpm		$ 25,900	1	$ 25,900
E1203	Motor	WME		50	$3,640	1	$ 3,640
E1300	Flotation (Cleaner)	WME	22.5 cu.ft		$ 20,200	4	$ 80,800
E1301	Motor	WME		7.5	$ 791	4	$3,164
E1400	Pan Filter	WME	0.2 tph	2	$ 10,800	1	$10,800
E1401	Motor			Included in E1400			
Subtotal							**$ 1,026,480**

Table 7.5 Total capital equipment cost for the placer processing plant.

Subtotal capital equipment cost	$3,008,109
Total capital equipment cost (2.15 × Subtotal)	**$6,467,434**

Table 7.6 Estimate for installed capital equipment cost.

1	Total Equipment Cost		$ 6,467,434
2	Miscellaneous Equipment Cost	(Item 1×0.1)	$ 646,743
3	Total Equipment Capital Cost (Add Item 1 and Item 2)		$ 7,114,117

	Construction Item	Factor	Cost = Factor × Item 3
4	Machinery Installation	0.26	$ 1,849,686
5	Foundation/Elevated Floors	0.42	$ 2,987,955
6	Structural/Steelwork	1.30	$ 9,248,431
7	Piping	0.78	$ 5,549,058
8	Electrical	0.67	$ 4,766,499
9	Instrumentation	0.05	$ 355,709
10	Roofing	0.13	$ 924,843

11	Subtotal (Item 3 to 10)	$ 32,796,358
12	Engineering(Item 11×0.10)	$ 3,279,636
13	**Total installed cost**	**$ 36,075,994**

Table 7.7 Estimate of labor requirements.

Employee classification	Source	Number of employee	Salary grade	Annual salary plus fringes at 1990 US $
Superintendent	CCCM	1	A	$ 77,500
General Forman	CCCM	1	B	$ 69,800
Maintenance Forman	CCCM	1	B	$ 69,800
Preparation Engineer	CCCM	3	C	$ 62,000
Shift Forman	CCCM	3	C	$ 62,000
Outside Forman	CCCM	1	C	$ 62,000
Assistance Shift Forman	CCCM	3	D	$ 54,800
Plant Operator	CCCM	3	E	$ 51,500
Mechanic	CCCM	10	E	$ 51,500
Electrician	CCCM	4	E	$ 51,500
Mobil Equipment Operator	CCCM	12	E	$ 51,500
Welder	CCCM	8	E	$ 51,500
Flotation Operator/Helper	CCCM	4	F	$ 45,500
Loadout operator	CCCM	1	F	$ 45,100
Sampler	CCCM	3	F	$ 45,100
Laborer	CCCM	4	G	$ 38,700
Laboratory Technician	CCCM	3	H	$ 32,200
Warehouseman	CCCM	3	H	$ 32,200
Clerk/Secretary	CCCM	1	I	$ 25,800

Table 7.8 Estimate of labor costs.

Labor category	Number of employees	Annual cost per employee at 1990 US $	Cost index	Total annual cost at 2006 US $
A	1	$ 77,500	1.32	$ 102,300
B	2	$ 69,800	1.32	$ 184,272
C	7	$ 62,000	1.32	$ 572,880
D	3	$ 54,800	1.32	$ 217,000
E	37	$ 51,500	1.32	$ 2,515,260
F	8	$ 45,100	1.32	$ 476,256
G	4	$ 38,000	1.32	$ 200,640
H	6	$ 32,000	1.32	$ 253,440
I	1	$ 25,000	1.32	$ 33,000
Subtotal	69			$ 4,555,048
Total annual labor costs (2.15 × Subtotal)				**$9,793,353**

Table 7.9 Estimate of froth flotation reagents costs.

Flotation reagents	**Source**	**Annual consumption (lb/yr)**	**$/lb**	**Annual cost**
Aero 350 Xanthate	Cytec	207,200	$ 1.00	$ 207,200
Aero 3477 Dithiophosphate	Cytec	103,100	$ 0.93	$ 95,883
Pine Oil	CCCM	256,800	$ 3.31	$ 850,008
Aero 70 frother (MIBC)	Cytec	128,400	$ 1.56	$ 200,304
Aero 65 frother	Cytec	154,700	$ 2.32	$358,904
Subtotal annual cost				$1,712,299
Total annual reagent costs (2.15×Subtotal)				**$3,681,443**

Table 7.10 Estimate of electric power costs.

Equipment #	Connected Hp	Diversity factor	Cost $/kWhr	Annual Cost
E101	200	0.85	0.25	$ 277,736
E201	500	0.85	0.25	$ 694,340
E400	7.5	0.85	0.25	$ 10,415
E403	25	0.85	0.25	$ 34717
E405	50	0.85	0.25	$ 69,434
E501	3	0.85	0.25	$ 4,166
E600	1.5	0.85	0.25	$ 2,083
E700	3	0.85	0.25	$ 4,166
E703	5	0.85	0.25	$ 6,943
E800	1	0.85	0.25	$ 2,777
E900	3	0.85	0.25	$ 4,166
E1002	2.5	0.85	0.25	$ 3,472
E1101	40	0.85	0.25	$ 55,547
E1201	640	0.85	0.25	$ 888,775
E1301	30	0.85	0.25	$ 41,660
E1400	2	0.85	0.25	$ 2,777
Subtotal				$ 2,418,627

Heating and Lighting 0.15 × Electric Motor Annual Cost	$ 315,473
Total annual electric power costs	**$2,418,627**

Table 7.11 Estimates of lubrication and vehicle fuel costs.

Feed rate	**Source**	**Annual tons of feed**	**Lubrication and vehicle fuel cost per ton**	**Annual cost**
1500 tph	CCCM	13,140,000	$ 0.06	$ 788,400
Total annual lubrication and vehicle fuel costs (2.15 × Subtotal)				**$1,695,060**

Table 7.12 Summary of annual operating costs.

Cost item	**Direct annual cost**
Labor	$ 9,793,353
Material and supplies	$ 711,412
Electric power	$ 2,418,627
Flotation reagents	$ 3,681,443
Flocculants	Considered but cost not included
Lubrication and vehicle fuel	$ 1,695,060
Subtotal	$18,299,895
Miscellaneous (5% of subtotal)	$ 914,995
Total cost	**$19,214,890**

Availability factors were applied to the following (Table 7.12) operating cost items: material and supplies, electric power, flotation reagents, lubrication and vehicle fuel. The labor cost was assumed unreduced by lower dredge and plant availability.

Table 7.13 Estimate of operating costs at 50% availability.

Cost item	**Direct annual cost**
Labor	$ 9,793,353
Material and supplies (0.5×Table 7.12)	$ 355,706
Electric power (0.5×Table 7.12)	$ 1,209,314
Flotation reagents (0.5×Table 7.12)	$ 1,840,722
Flocculants	Considered but cost not included
Lubrication and vehicle fuel(0.5×Table 7.12)	$ 847,530
Subtotal	$ 14,046,624
Miscellaneous (5% of subtotal)	$ 702,331
Total cost	**$ 14,748,955**

Table 7.14 Estimate of operating costs at 60% availability.

Cost item	Direct annual cost
Labor	$ 9,793,353
Material and supplies (0.6× Table 7.12)	$ 426,847
Electric power (0.6×Table 7.12)	$ 1,415,176
Flotation reagents (0.6×Table 7.12)	$ 2,208,866
Flocculants	Considered but cost not included
Lubrication and vehicle fuel (0.6×Table 7.12)	$ 1,017,036
Subtotal	$ 14,897,276
Miscellaneous (5% of subtotal)	$ 744,864
Total cost	**$ 15,642,142**

Table 7.15 Estimate of operating costs at 70% availability.

Cost item	Direct annual cost
Labor	$ 9,763,353
Material and supplies (0.7×Table 7.12)	$ 497,988
Electric power (0.7×Table 7.12)	$ 1,693,039
Flotation reagents (0.7×Table 7.12)	$ 2,577,010
Flocculants	Considered but cost not included
Lubrication and vehicle fuel (0.7×Table 7.12)	$ 1,186,542
Subtotal	$15,747,932
Miscellaneous (5% of subtotal)	$ 787,397
Total cost	**$ 16,535,329**

Table 7.16 Estimate of operating costs at 80% availability.

Cost item	Direct annual cost
Labor	$ 9,793,353
Material and supplies (0.8×Table 7.12)	$ 569,130
Electric power (0.8×Table 7.12)	$ 1,934,902
Flotation reagents (0.8×Table 7.12)	$ 2,945,154
Flocculants	Considered but cost not included
Lubrication and vehicle fuel (0.8×Table 7.11)	$ 1,356,048
Subtotal	$ 16,598,587
Miscellaneous (5% of subtotal)	$ 829,929
Total cost	**$ 17,428,516**

Table 7.17 Estimate of operating costs at 90% availability.

Cost item	Direct annual cost
Labor	$ 9,793,353
Material and supplies (0.9×Table 7.12)	$ 640,271
Electric power (0.9× Table 7.12)	$ 2,176,764
Flotation reagents (0.9× Table 7.12)	$ 3,313,299
Flocculants	Considered but cost not included
Lubrication and vehicle fuel (0.9× Table 7.12)	$ 1,525,554
Subtotal	$17,449,241
Miscellaneous (5% of subtotal)	$ 872,462
Total cost	**$ 18,321,703**

7.1 Economic analysis and evaluation

By employing accepted engineering economic evaluation methods[55], the cost per ton to process the marine sediments of Southwestern Alaska was estimated based on the capital and operating costs for the designed flowsheet. 60% of the total installed capital equipment cost was assumed to be invested at time zero, while 40% was assumed to be invested at the end of year one. Full production is assumed during year one; processing 1500 tons per hour of placer ore. The costs were annualized assuming a 15% minimum rate of return (MROR) and a 9 year of production life. As an example of the economic

evaluation procedure, consider the cost of production estimate shown in Table 7.18 and the project timeline presented in Figure 7.4.

Table 7.18 Service producing analysis.

Year	0	1	2	.....	10
Capital Cost	$ 21,645,890	$ 14,430,398			
Labor			$ 9,793,353	....	$ 9,793,353
Material and Supplies			$ 711,412	....	$ 711,412
Electric Power			$ 2,418,627	...	$ 2,418,627
Flotation Reagents			$ 3,681,443	....	$ 3,681,443
Lubrication and Vehicle Fuel			$ 1,695,060	...	$ 1,695,060
Miscellaneous			$ 914,995	....	$ 914,995
Total	$ 21,645,596	$ 14,430,398	$ 19,214,890	...	$ 19,214,890

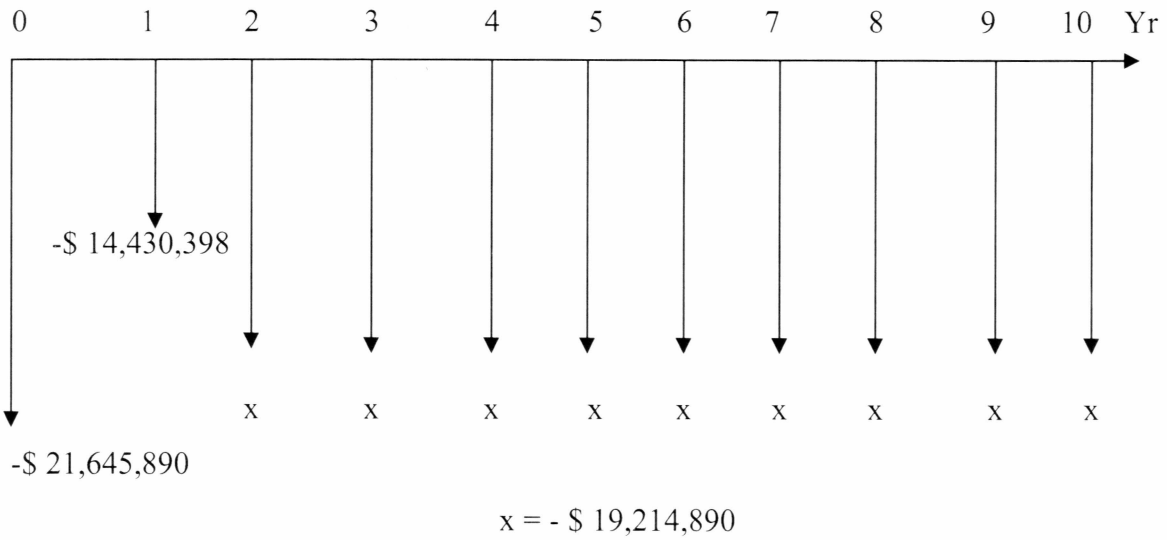


Figure 7.2 Cash flow time line diagram.

The capital costs are time zero and year 1 and operating costs were converted to a uniform series of constant, end-of-year costs by employing engineering economic analysis concepts; the single payment compound amount factor $(F/P_{i,n})$, the uniform series present worth factor $(P/A_{i,n})$, the single payment present worth factor $(P/F_{i,n})$,and the capital recovery factor $(A/P_{i,n})$, defined mathematically as follow:

$$F/P_{i,n} = (1+i)^n \qquad [7.1]$$

$$P/A_{i,n} = \frac{\left[(1+i)^n - 1\right]}{i(1+i)^n} \qquad [7.2]$$

$$P/F_{i,n} = \frac{1}{(1+i)^n} \qquad [7.3]$$

$$A/P_{i,n} = \frac{i(1+i)^n}{[(1+i)^n - 1]} \qquad [7.4]$$

where;

P = present worth or cost

F = future worth or cost

A = uniform series of equal end-of-year payments or costs

i = the minimum rate of return (MROR)

n = the number of compounding periods over the life of the investment

Net Year 1 Value = $21,645,890(F/P_{15\%,1}) + 14,430,398 + 19,214,890(P/A_{15\%,9})$

Net Year 1 Value = $\$131,061,627$

Annual Cost = $\$131,061,627(A/P_{15\%,9})$

Annual Cost = $\$27,457,154$ per year (year 2-year 10)

Tons of Ore Mined = $1500\,ton/hr \times 24hr/day \times 365\,days/year \times 1.0\,availabilty$

Tons of Ore Mined = $13,140,000\,TPY$

Annual Cost per ton = $\frac{\$27,457,627/yr}{13,140,000\,TPY} = \$2.09/ton$

With this methodology in mind and using operating costs in Tables 7.12- 7.17, six different scenarios were considered in estimating the cost per ton to process the marine sediments. However, these scenarios also consider that operating costs escalate at a 3% annual rate over the life of production. Tables 7.19-7.23 show the results of the economic analysis.

Table 7.19 Service producing analysis at 50% availability.

Years	0	1	2	3	4	5	6	7	8	9	10
Capital Cost	21,646	14,430									
Operating Costs			14,749	15,191	15,647	16,117	16,600	17,098	17,611	18,139	18,684
Total	21,646	14,430	14,749	15,191	15,647	16,117	16,600	17,098	17,611	18,139	18,684

Capital costs and operating costs are stated in thousands (1000) of US dollars.

By using equations 7.1, 7.3, and 7.4, the annual cost per ton was calculated as follows;

$$\text{Net Year 1 Value} = \left[\begin{array}{l}[21,646(F/P_{15\%,1}) + 14,430 + 14,749(P/F_{15\%,1}) + 15,191(P/F_{15\%,2}) + 15,647(P/F_{15\%,3}) + 16,117(P/F_{15\%,4}) \\ +16,600(P/F_{15\%,5}) + 17,098(P/F_{15\%,6}) + 17,611(P/F_{15\%,7}) + 18,139(P/F_{15\%,8}) + 18,684(P/F_{15\%,9})] \times \$1000\end{array}\right]$$

$$\text{Net Year 1 Value} = \$116,645,063$$

$$\text{Annual Cost} = \$116,645,063(A/P_{15\%,9}) = \$24,445,306 \; per \; year$$

$$\text{Tons of Ore Mined} = 1500 \, ton/hr \times 24 hr/day \times 365 \, days/year \times 0.5 \, availabilty = 6,570,000 \; TPY$$

$$\text{Annual Cost per ton} = \frac{\$24,445,306/yr}{6,570,000 TPY} = \$3.72/ton$$

Table 7.20 Service producing analysis at 60% availability.

Years	0	1	2	3	4	5	6	7	8	9	10
Capital Cost	21,646	14,430									
Operating Costs			15,642	16,111	16,595	17,092	17,605	18,133	18,677	19,238	19,815
Total	21,646	14,430	15,642	16,111	16,595	17,092	17,605	18,133	18,677	19,238	19,8151

Capital costs and operating costs are stated in thousands (1000) of US dollars.

Using equations 7.1, 7.3, and equation 7.5, the annual cost per ton was calculated as follows;

$$\text{Net Year 1 Value} = \left[\begin{array}{l}[21,646(F/P_{15\%,1}) + 14,430 + 15,642(P/F_{15\%,1}) + 16,111(P/F_{15\%,2}) + 16,595(P/F_{15\%,3}) + 17,092(P/F_{15\%,4}) \\ +17,605(P/F_{15\%,5}) + 18,133(P/F_{15\%,6}) + 18,677(P/F_{15\%,7}) + 19,238(P/F_{15\%,8}) + 19,815(P/F_{15\%,9})] \times \$1000\end{array}\right]$$

$$\text{Net Year 1 Value} = \$121,326,648$$

$$\text{Annual Cost} = \$121,326,648(A/P_{15\%,9}) = \$25,426,426 \, per \ year$$

$$\text{Tons of Ore Mined} = 1500 \, ton/hr \times 24 hr/day \times 365 \, days/year \times 0.6 \, availabilty = 7,884,000 \ TPY$$

$$\text{Annual Cost per ton} = \frac{\$25,426,426/yr}{7,884,000 \, TPY} = \$3.23/ton.$$

Table 7.21 Service producing analysis at 70% availability.

Years	0	1	2	3	4	5	6	7	8	9	10
Capital Cost	21,646	14,430									
Operating Costs			16,535	17,031	17,542	18,068	18,610	19,169	19,744	20,336	20,946
Total	21,646	14,430	16,535	17,031	17,542	18,068	18,610	19,169	19,744	20,336	20,946

Capital costs and operating costs are stated in thousands (1000) of US dollars.

Using equations 7.1, 7.3, and equation 7.4, the annual cost per ton was calculated as follows.

$$\text{Net Year 1 Value} = \begin{bmatrix} [21,646(F/P_{15\%,1}) + 14,430 + 16,535(P/F_{15\%,1}) + 16,17,031(P/F_{15\%,2}) + 17,542(P/F_{15\%,3}) + 18,068(P/F_{15\%,4}) \\ +18,610(P/F_{15\%,5}) + 19,169(P/F_{15\%,6}) + 19,744(P/F_{15\%,7}) + 20,336(P/F_{15\%,8}) + 20,946(P/F_{15\%,9})] \times \$1000 \end{bmatrix}$$

$$\text{Net Year 1 Value} = \$126,008,232$$

$$\text{Annual Cost} = \$126,008,232(A/P_{15\%,9}) = \$26,407,545 \; per \; year$$

$$\text{Tons of Ore Mined} = 1500 \, ton/hr \times 24 hr/day \times 365 \, days/year \times 0.7 \, availabilty = 9,198,000 \; TPY$$

$$\text{Annual Cost per ton} = \frac{\$26,008,232/yr}{9,198,000 TPY} = \$2.87/ton$$

Table 7.22 Service producing analysis at 80% availability.

Years	0	1	2	3	4	5	6	7	8	9	10
Capital Cost	21,646	14,430									
Operating Costs			17,429	17,952	18,490	19,045	19,616	20,205	20,811	21,435	22,079
Total	21,646	14,430	17,429	17,952	18,490	19,045	19,616	20,205	20,811	21,435	22,079

Capital costs and operating costs are stated in thousands (1000) of US dollars.

$$\text{Net Year 1 Value} = \left[\begin{array}{l}[21,646(F/P_{15\%,1}) + 14,430 + 17,429(P/F_{15\%,1}) + 17,952(P/F_{15\%,2}) + 18,490(P/F_{15\%,3}) + 19,045(P/F_{15\%,4}) \\ +19,616(P/F_{15\%,5}) + 20,205(P/F_{15\%,6}) + 20,811(P/F_{15\%,7}) + 21,435(P/F_{15\%,8}) + 22,079(P/F_{15\%,9})] \times \$1000\end{array}\right]$$

$$\text{Net Year 1 Value} = \$130,695,059$$

$$\text{Annual Cost} = \$130,695,059(A/P_{15\%,9}) = \$27,389,764 \ \textit{per year}$$

$$\text{Tons of Ore Mined} = 1500 \textit{ton/hr} \times 24 \textit{hr/day} \times 365 \textit{days/year} \times 0.8 \textit{availabilty} = 10,512,000 \ \textit{TPY}$$

$$\text{Annual Cost per ton} = \frac{\$27,389,764/yr}{10,512,000 TPY} = \$2.61/ton$$

Table 7.23 Service producing analysis at 90% availability.

Years	0	1	2	3	4	5	6	7	8	9	10
Capital Cost	21,646	14,430									
Operating Costs			18,322	18,872	19,438	20,021	20,622	21,877	21,877	22,534	23,210
Total	21,646	14,430	18,322	18,872	19,438	20,021	20,622	21,877	21,877	22,534	23,210

Capital costs and operating costs are stated in thousands (1000) of US dollars.

$$\text{Net Year 1 Value} = \left[\begin{array}{l}[21,646(F/P_{15\%,1}) + 14,430 + 18,322(P/F_{15\%,1}) + 18,872(P/F_{15\%,2}) + 19,438(P/F_{15\%,3}) + 20,021(P/F_{15\%,4}) \\ +20,622(P/F_{15\%,5}) + 21,877(P/F_{15\%,6}) + 21,877(P/F_{15\%,7}) + 22,534(P/F_{15\%,8}) + 23,210(P/F_{15\%,9})] \times \$1000\end{array}\right]$$

$$\text{Net Year 1 Value} = \$135,376,644$$

$$\text{Annual Cost} = \$135,376,644(A/P_{15\%,9}) = \$28,370,883$$

$$\text{Tons of Ore Mined} = 1500\,ton/hr \times 24hr/day \times 365\,days/year \times 0.9\,availabilty = 11,826,000\ TPY$$

$$\text{Annual Cost per ton} = \frac{\$28,370,883/yr}{11,826,000\,TPY} = \$2.40/ton$$

The total installed capital equipment cost is estimated to be $36 million. The annualized costs per ton to process the placer ore are estimated in Table 7.18 to Table 7.23; in the range of $2.40 to $3.72 depending upon plant availability, 90% to 50%, respectively. Based on the data presented Hartman and Mutmansky [56], and adjusted using WME cost indexes and the location factor, large scale (1500 tph) dredging cost for the Goodnews Bay, Alaska area can be estimated at $2.50 - $3.00 per ton. This estimate can be combined with the processing cost estimates above, in order to estimate the break even grade necessary for production. Current gold and PGM prices are presented in Table 7.24.

Table 7.24 June 2007 precious metals prices.

Metal	US dollar/ounce
Gold	650
Platinum	1275
Palladium	370
Rhodium	6000
Iridium	400
Ruthenium	350
Osmium	360

These precious metal prices, and the overall costs of mining and processing estimated in this thesis, indicate that very low grade marine sediments could be developed and brought into production from the offshore environment near Goodnews Bay, Alaska. The cost estimation and economic analysis are only intended to provide estimates to aid future development decisions.

The pan concentrate used in this study is not representative of the marine sediments at Goodnews Bay, Alaska. Therefore these test results should be evaluated considering what they represent, and not be depended upon for design, without additional test work and extensive sampling of the marine sediments.

This study assumed a high dredge capacity based on the assumption that large tonnage, low-grade, PGM and gold resources would be the target for mining and processing at Goodnews Bay, Alaska. Perhaps the cost estimates developed here could be used to estimate costs for other mining and processing feed scenarios at Goodnews Bay, Alaska, based on the equation presented by Kelly and Spottiswood[57]

$$C_n = C_e \left(\frac{I_n}{I_e} \right)^n \left(\frac{CI_n}{CI_e} \right) K_{Lo} \qquad [7.5]$$

where:

C_n, C_e = cost of new and existing plants respectively.

CI_n, CI_e = cost indexes at the time of construction of new and existing plants, respectively.

I_n, I_e = capacities of new and existing plants, respectively.

n = capacity exponent.

K_{Lo} = location factor.

As noted by the authors, Kelly and Spottiswood, "the accuracy of the method depends significantly on the comparability of the plants. Ideally the plant should use not only the same process, but the same flowsheet, and if not the reliability can be expected to fall off rapidly"[57]. The difficulty of finding reliable and proven values of the exponent 'n' could also limit the use of the equation 7.5 for applying this study's cost estimates to higher or lower feed rates.

Chapter 8 Summary and Conclusions

Froth Flotation is a promising processing technique to beneficiate the fine and/or flaky placer platinum and placer gold found in the marine Sediments in southwestern Alaska. The various metallurgical laboratory tests used to study the pan concentrate have shown that the placer sediments contain appreciable amounts of gold and platinum. The low intensity, magnetic separation demonstrated that 97% and 100% of placer platinum and placer gold reported to the non-magnetic material, respectively. Results of particle size analysis indicate that the placer gold is coarser than placer platinum. It also showed that majority of the placer gold exist between 50 mesh size fraction and 150 mesh size fraction, while the majority of the platinum exist between 100 mesh size fraction and 200 mesh size fraction.

The results of flotation studies showed that the investigated collectors all provided high gold recoveries, 82.7%-99.8%. However, their effectiveness as collectors for the platinum mineralogy varied drastically, yielding platinum recoveries over a broad range, 4.1%-80.4%. Individually, Aero 350 (potassium amyl xanthate) produced the highest recovery of placer gold and placer platinum. A combination of Aero 350 and Aero 3477 (sodium isobutyl dithiophosphate) gave 75% placer platinum recovery, likely due to presence of the xanthate. The dithiophosphate demonstrated excellent froth stabilization characteristics. Qualitative and quantitative microprobe analysis of PGM grains from the flotation concentrate showed grains of nearly pure iridium, isoferroplatinum and Pt-Rh-Ir-Fe-S-As mineralogy.

Future precious metals prices and exploration programs will dictate if the 1500 tph flowsheet develop during this research demonstrates suitable economics. Estimated processing costs at a 1500 tph feed rate are in the range of $2.40 to $3.72 per ton, depending upon plant availability and conditions at Goodnews Bay.

8.1 Recommendation for future work

After a critical examination of all the results and reviewing all work that has been done on the marine sediments at Goodnews Bay, the investigator recommends the following work to be considered in the future:

1. A detailed study of the mineralogy of the Goodnews Bay, Alaska marine sediments is required for improving the recovery of PGM. Reflected light microscopy and microprobe analysis should be carried out.
2. Different types of xanthate reagents should be investigated. Potassium amyl xanthate should be combined with other reagents to investigate their performance on the placer platinum and placer gold.
3. Dosage optimization for potassium amyl xanthate and other promising reagents should be conducted.
4. The latest trend in the recovery of liberated particles from ore is to increase the power rating of the flotation cell, increasing the turbulent conditions to enable more effective particle-bubble attachment. It is also suggested that smaller bubbles are required to float small particles more efficiently[58]. Bubble size and

froth stability issues relating to the low metals content of marine sands should be conducted in future studies.

5. Flotation tests should also be conducted with saltwater, though literature suggests salt water effects should not be detrimental to precious metals recovery[35].

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